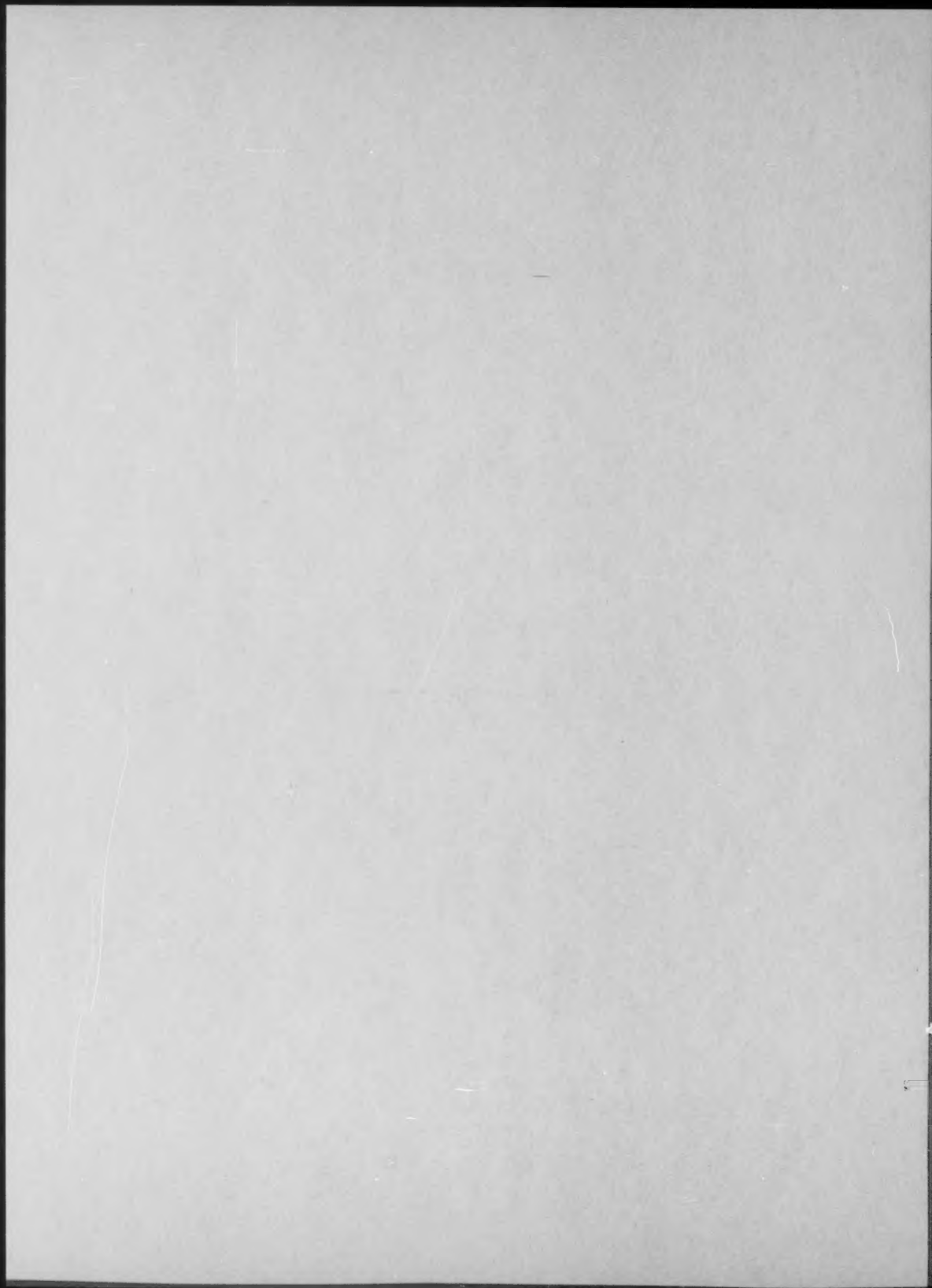


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## MORE METAL

In 1956 the industrial undertakings of the Ministry of Iron and Steel of the U.S.S.R. achieved a further advance in output. As compared with 1955 there was an increase in pig-iron smelting by 7.8%, steel production by 7.8%, the output of rolled sections by 7.1%, steel tubes by 8.5%, coke firing by 6.9% and iron-ore extraction by 8.4%.

This increased production was achieved largely on the basis of the higher productivity of existing plants. The coefficient of utilization of the blast furnace capacity was improved by 2.6%, reaching 0.78, and steel tapping per square meter of hearth of the open-hearth furnaces rose by 6%, reaching 7.19 t/m<sup>2</sup>. The utilization of rolling mills and tube plant has been improved.

Despite the increased production, however, the 1956 plan was not fulfilled by the plants as a whole. Under-fulfilment of the plan was caused by serious shortcomings in the utilization of manufacturing capacities and by delays in bringing new plants into production.

A large quantity of pig-iron was lost through unsatisfactory preparation and neutralization of the iron-ore at the mines and at a number of plants.

While improved performances were recorded at some blast furnace plants as compared with 1955, namely Magnitogorsk, Kuznetsk, Novo-Tagil, "Azovstal", Kriivorozh, "Zaporozhstal", Petrovsk and others, the performance of other plants showed a decline namely Dzerzhinsky, Enakiev, Makeeva, Kramatorsk, Chelyabinsk Metallurgical, Ashinsk, Kosogorsk and Nizhny-Saldinsk.

In individual concerns there were long stoppages of the blast furnaces. During eleven months of 1956 blast furnace stoppages at the Dzerzhinsky plant amounted to 2.7%, at the Orsko-Khalilovo 1.9%, at the Chelyabinsk Metallurgical 1.5%, while at the Petrovsk plant stoppages amounted to 0.4% and in the Magnitogorsk combine and at the "Azovstal" plant, 0.5%.

In steel smelting production there was considerable short fall through long stoppages on open-hearth furnaces for cold and hot repairs. During eleven months in 1956 at the Makeevka plant stoppages amounted to 14.3%, at the Enakiev plant 13.7%, the Stalin plant 13.2%, at the Serov plant 12.2%, while at the Kuznetsk and Magnitogorsk combines the open-hearth furnaces were idle to the extent of 6.7 and 7.8%.

Poorer performances than in 1955 were recorded for the open-hearth furnaces at the Dzerzhinsky, Stalin, Taganrog, Vyksun, Chelyabinsk Metallurgical, Omutninsk, Kazakh, and Ashinsk plants. Improved open-hearth furnace performances were achieved by the Kuznetsk and Magnitogorsk combines, and by the Novo-Tagil, Voroshilov, "Zaporozhstal" and Makeevka plants together with some others.

One of the main causes of unsatisfactory performance in the steel mills exists in the lengthy idle periods on the rolling mills. During eleven months in 1956 stoppages at the Novo-Tagil plant amounted to 23.1%, on the Ashinsk plant 22.3%, Kramatorsk 21.7%, and Voroshilov 18.7%. At several steel mills the repair work is still poorly organized the production program is badly tackled and the orders are not distributed to the best effect, which results in excessive stoppages for roll changing.

In 1957, the second year of the sixth Five-Year Plan the most important task for every worker in the iron and steel industry is the maximum utilization of the reserves available in the undertakings.

The plan envisages for 1957 an increase in production as compared with 1956 amounting to 2.4 million tons of pig-iron, 2.4 million tons of steel and 2.1 million tons of rolled sections.

Blast-furnace operators must give serious attention to the preparation of the raw material and neutralization of the ore in order to provide constant uniformity of the charge. There is also an urgent need for a higher quality agglomerate, and greater production of fluxing agglomerate with higher basicity.

One of the main tasks of the steel industry is the conversion of all open-hearth furnaces to the basic process, for which purpose it is necessary at the earliest possible date to complete the construction of new manufacturing capacities for the output of chrome magnesite brick at the Satka factory.

It is necessary to reduce considerably the stoppage times at open-hearth furnaces, to improve the production program and the repair organization, and to increase the use of oxygen.

The workers of the rolling mills and tube mills are confronted with the task of achieving an all-around improvement in the maintenance of equipment and in the repair organization, of increasing the rolling rates, of better application of the experience of the leading brigades and of individual rollers. A sharp increase in the output of rolled section runs is essential.

One very important task is the further reduction in the cost of metal. We must manufacture cheap metal, achieve the greatest economy in the utilization of material, fuel and electrical energy, and reduce the delays in transport. We must achieve an all-around improvement in the labor return on each undertaking.

Substantial tasks confront the steel workers in 1957. Let us turn all our efforts to the honorable fulfillment of the assignments of the party and government.

## BLAST-FURNACE PRODUCTION

### BLAST-FURNACE OPERATION WITH DISTURBED PROGRAM

L. Ya. Shparber

(Head of the Blast-Furnace Technological Group of the Magnitogorsk Metallurgical Combine)

Statistics relating to disturbances in the operation of blast furnaces of the Magnitogorsk metallurgical combine over the last five years show that in the majority of cases (82%) the disturbance was caused by a variation in the quality of the raw material and coke. To a much smaller degree (18%) the disturbances were associated with insufficiently skilled foremen or inadequacy of the electrical and mechanical equipment.

We will study three types of disturbances in blast-furnace operation: 1) the result of over-blowing with an excess of fine particles in the charge, 2) the result of variation in the charging program and 3) results of divergent influences of the blast-furnace foremen.

#### Disturbance in Blast-Furnace Operation as the Result of Over-Blowing with an Excess of Particles in the Charge.

The blast-furnace performance before the disturbance was characterized by the following mean monthly indices:

Coefficient of utilization of available volume	0.610
Coke consumption per 1 t pig-iron, kg	0.688
Blast-furnace dust carried off per 1 t pig-iron, %	4.7
Ore charge	2.59
Slag output per 1 t pig-iron, kg	478
Average CO <sub>2</sub> content in the gas, %	13.8
Blast-furnace gas pressure, atm	0.8
Blast:	
Quantity (measured at the blower intake), m <sup>3</sup> /min	2800
Pressure, atm	2.0
Temperature, °C	907
Throat temperature, °C	404
Number of sets and break-outs of the charge	none
Furnace stoppages, %	0.32

The burden and charging system for this period are shown in the daily log sheet.

There was an opinion among the foremen that the activity in the center of the furnace should be intensified by increasing the blast, and therefore, on one of the shifts 100 m<sup>3</sup>/min of air was added in stages of 50 m<sup>3</sup>/min. After increasing the blast the gas flow became unstable and on charging the agglomerate with a high content of fine fractions there was a tendency to chimneying.

The unstable gas flow caused the blast-furnace foremen to maintain a high furnace heat, as indicated by the data below relating to the silica content in the pig-iron during the five days of furnace operation after changing over to increased blast:

Si content in the pig-iron, %:	below 0.8	0.8-0.9	0.9-1.0	over 1.0
Number of cases,	none	2	15	16

On operating the furnace at a high heat rate the basicity of the slag reached 1.17 providing the following sulfur content in the pig-iron:

S content in the pig-iron, %:	Below 0.02	0.02-0.03	0.03-0.04	0.040-0.044	over 0.044
Number of cases	4	27	3	1	none

With the high heat rate the humidity of the blast was raised to 34 g/m<sup>3</sup>. In this way, during the five days of blast furnace operation the fluctuations in the heat basicity of the slag and humidity ranged: according to silicon content from 0.82 to 1.17%, according to basicity of the slag from 1.05 to 1.17, and for humidity of the blast from 20 to 34 g/m<sup>3</sup>.

With normal distribution of the gas flow the basicity of the slag, on average 1.12, could not be excessive; under the conditions brought about by chimneying, then, the decision to raise the humidity of the blast for the given basicity of the slag was incorrect.

An analysis was made directly of the disturbed furnace operation. An assessment can be made of performance over this period of days from the daily log sheet and Fig. 1 and 2.

The data show that the furnace heat was adequate, but unstable. Thus, the silicon content at the end of the day varied from 0.78% to 1.14%. The gas flow was also unstable, and all the signs indicated a tendency to chimneying.

As is seen from the charts, chimneying clearly discernible, began at 13 hours. At the same time the slag temperature points rose by 40°, the blast furnace gas temperature curves became scattered and there were frequent dips of the indicator rod.

While during the first half of the second shift thirty-nine charges were fed into the furnace corresponding to 261 t of fired coke, in the second half of the shift thirty-three charges were fed (221 t of coke); hence, the difference in the rate of coke firing between the first and second halves of the shift was 40 t. Chimneying led to break-outs of the charge, which commenced at 14 h 30.\*

The nature of the disturbance indicates that at 16 h a large part of the furnace cross section was not operating and the gas, forming a broad channel, passed out of the furnace at a temperature of 540-560°. Comparison of the meter readings with the operation of the tuyeres establishes that the channel developed at tuyeres numbers 6, 7 and 8. Incomplete charging of the furnace, allowed under these circumstances, continued for five hours thirty minutes until the furnace was shut down for changing nozzles numbers 7 and 8, which were blocked with slag. The nature of the temperature curve for the gases at the gas outlet (single line) shows that with an incompletely charged furnace the gases intermingled above the stock line and passed out of the furnace at approximately uniform temperature.

A number of measures were adopted for equalizing the gas flow. At 16 h 30 the furnace was put on to a reduced pressure, and at 18 h 15 a smelt was made for equalizing the gas flow. At 20 h and 20 h 30 the blast was reduced in stages by 100 m<sup>3</sup>/min. Further coke was charged into the furnace only at 19 h. All these measures, however, were adopted after long delays.

Hence, disruption in the operation of the blast furnace occurred as the result of: 1) excessive increase in the blast with a high content of fines in the charge, and 2) excessive humidity of the blast with basic slag, damping the activity of the furnace.

Disturbance in the blast-furnace performance could be avoided by taking correct steps for regulating the furnace operation. It would be necessary first not to allow excessive increase in the blast with a high content in the charge of fine fractions; second, with basic slag, chimneying and a high heat rate to reduce the humidity of the blast and, in accordance with the heat, to lower the temperature of the hot blast; third, the foreman taking over the shift at 16 h should at 17 h (commencing tapping at 16 h 30) change the gas flow. For this purpose under the given conditions (with adequate heat) it would be necessary to reduce the pressure of the blast sharply to 0.2 atm by changing over to the smelt, to hold at this pressure for approximately five minutes, to charge up the furnace with lumpy material and, having topped up, to introduce air, setting the blast at 150 m<sup>3</sup>/min less than previously. The basicity of the charge should be dropped to 1.05 for two shifts.

\* The daily log sheets in the original were not reproducible. - Publisher's note.

\*\* 14 h 30 = 14 hours, 30 minutes.

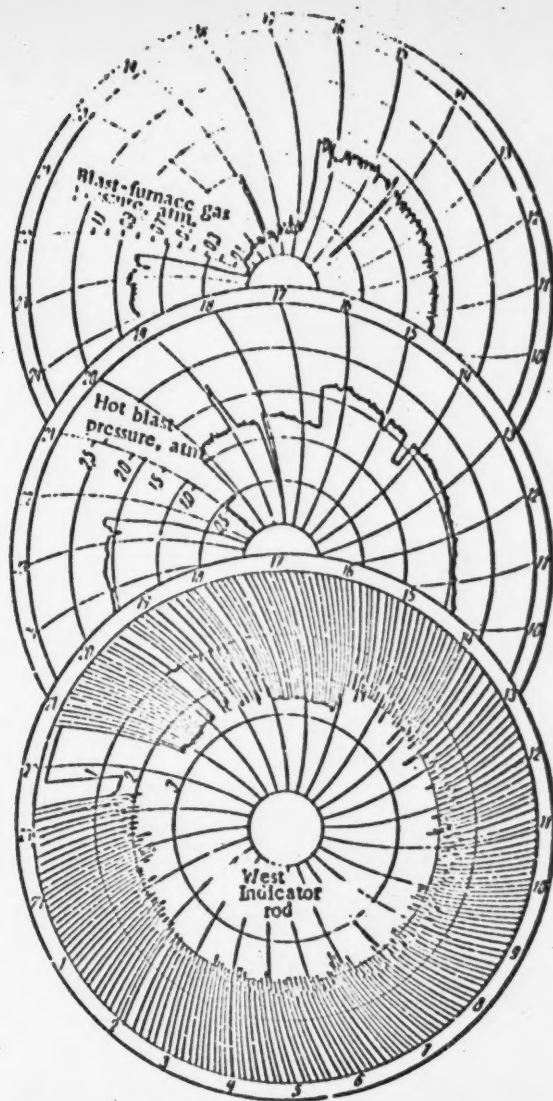


Fig. 1. Recorder charts of blast furnace gas pressure, hot-blast pressure and height of burden.

Disturbed Furnace Performance Connected with a Variation in the Slag Conditions.

Performance indices are set out below for a blast furnace together with the  $\text{CO}_2$  content across the radius of the throat during the day preceding the disturbance; the pig-iron and slag analyses are set out in the table.

### Performance Indices of the Blast Furnace :

Quantity of ore smelted, t/day	4318
Quantity of coke fired, t/day	1600.2
Limestone consumed, t/day	33.8
Blast:	
Temperature °C	900
Humidity, g/m <sup>3</sup> air	14
Quantity (measured at the blower intake) m <sup>3</sup> /min	2700
Pressure of the blast furnace gas, atm	0.9
Temperature of the throat, °C	362
Ore charge	2.68
CO <sub>2</sub> content across the radius of the throat :	
CO <sub>2</sub> content %:	14.8      17.2      19.5      19.7      18.7      11.8
Distance from the periphery, mm:	50      400      850      1400      2200      3300

### Daily Pig-Iron and Slag Analyses

Content %						basicity (CaO : SiO <sub>2</sub> )
Si	Mn	S	SiO <sub>2</sub>	CaO	MgO	
0.51	0.17	0.035	35.9	42.15	6.60	1.17
0.50	0.16	0.021				
0.57	0.18	0.021				
0.73	0.17	0.030				
0.64	0.16	0.032	35.20	42.85	6.60	1.22
0.56	0.15	0.021				
0.60	0.18	0.021				
0.56	0.16	0.037				

The overall CO<sub>2</sub> content (sampling point at the dust arrestor) was 15.2% in the first shift and 17.6% in the third shift.

It can be seen from these data that the furnace was being intensely operated and that the heat in the furnace was sufficient to render slag of the given composition liquid (200 kg of limestone was delivered in the charge). Limestone was left out from the first charge in the second shift since the foreman considered that the slag had a high basicity. In all, one hundred and ninety-nine charges were fed without limestone. This led to a reduction of the basicity of the slag (the CaO : SiO<sub>2</sub> ratio fell to 1.05), which reduced the heat of the furnace.

According to our concepts the slag in the blast furnace must absorb the heat moving up through the furnace to the throat, flow off through the hearth, yielding up its heat, and after combining with the coke ash run off from the furnace. With optimum slag conditions the zone of moderate temperatures will be at the maximum and the furnace will heat right through. If the slag does not carry out this function the moderate temperature zones will be confined and the high temperature zones will be displaced toward the stack. In this case melting will occur not in the bosh but in the blast-furnace stack and the quantity of heat delivered to the furnace by the slag will be reduced.

With such a reduced furnace heat in the first shift on the 7th June, 150 kg of limestone was included in the charge from the twenty-ninth charge.

Measures for raising the heat were not applied until the beginning of the second shift. At 8 h 30 the humidity of the blast was reduced to 13.5 g/m<sup>3</sup>, at 10 h 15 to 12 g/m<sup>3</sup> and at 13 h to 8 g/m<sup>3</sup> air.

However, the furnace heat was inadequate and with the addition of the charge with increased lime content it was necessary to raise the pressure of the hot blast (charts Figs. 3 and 4), the descent of the charge was

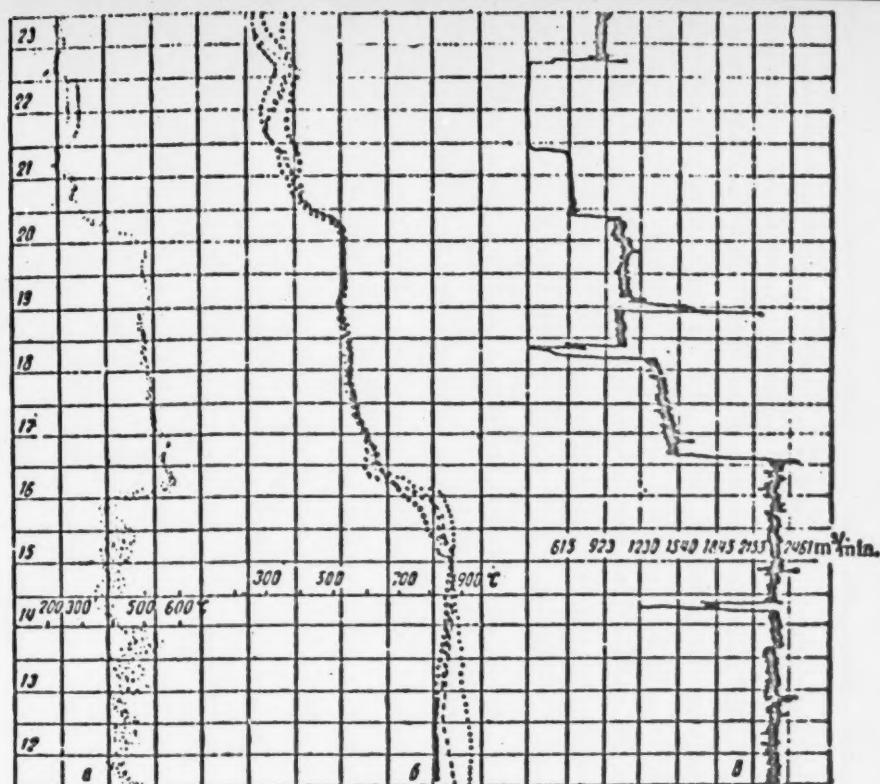


Fig. 2. Recorder charts (a) blast-furnace gas temperature (b) periphery gas temperature, and (c) air rate.

slowed down (time 9 h 30 to 10 h 30, Fig. 3) and at 10 h 30 the first chimney was formed in the column of material. The occurrence of this chimney was recorded by the meters showing an increased air flow; the formation of the channel was indicated by a saw-tooth dispersion of the coal-gas temperature points, a temperature drop in the shaft brick-work at point number 2 and dips in the indicator rods (charts, Figs. 3 and 4).

After the furnace had been charged up and six charges had been slowly melted through, a second wider channel was formed in the column of material; the coal-gas temperature rose sharply at the same time from 370 to 950°. The origin both of the first and second channel was found in the slag zone, as indicated by the furnace-gas pressure charts. If the channels had originated in the dry zone then the peaks indicating the pressure rise would be considerably greater with the formation of such a broad channel.

The formation of both chimneys could have been avoided if the foreman had simultaneously reduced the blast. This measure, however, was adopted too late and the blast was reduced only at 12 h 15 by 100 m<sup>3</sup>/min and again after 15 minutes by 100 m<sup>3</sup>/min.

At this high blast-furnace temperature it was necessary to take off the high pressure from the furnace and to bring in the snort. This could not be done immediately since the slag began to block the tuyeres, and therefore, the pressure of the hot blast was reduced gradually, allowing the slag to run into the hearth. In order to raise the heat and to break up the charge, individual skips and dead charges at 73 t coke were fed into the furnace, as the result of which the silicon content in the pig-iron was raised only to 1.16%. The furnace was brought back to normal operation only in the first shift of the 8th June.

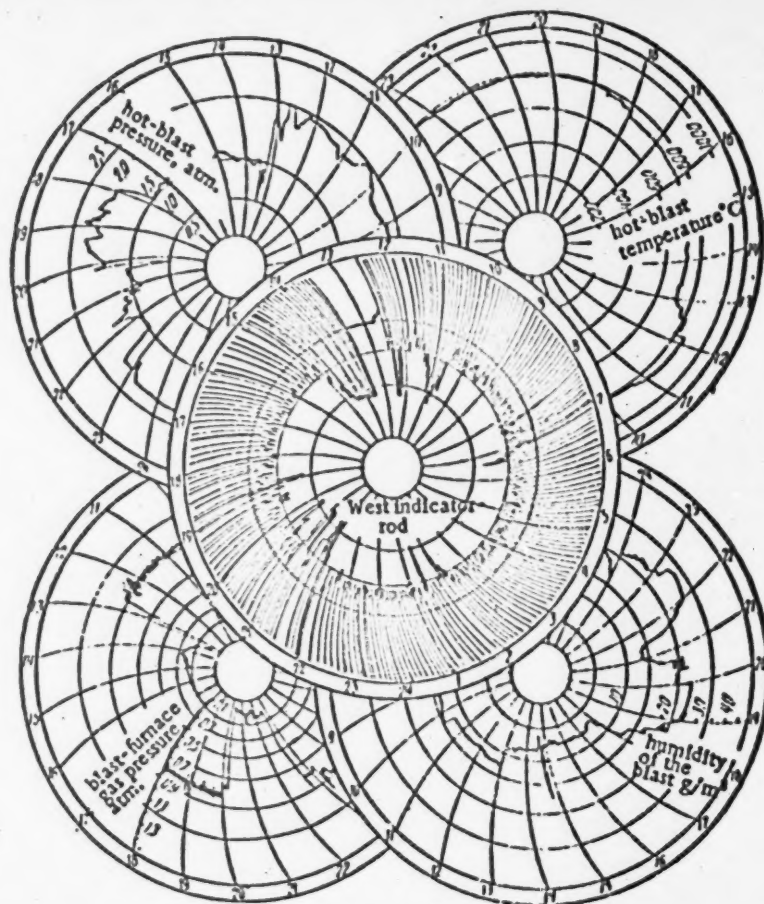


Fig. 3. Recorder charts for hot blast pressure and temperature, humidity of the blast, blast-furnace gas pressure and height of burden.

#### Disturbed Furnace Operation Associated with Divergent Influences of the Foremen Working on the One Furnace

A blast furnace can operate smoothly only when the three foremen operating the one furnace agree as to the manner of operation. Even for an ideal set of conditions any deviation from this elementary rule leads to disturbed performance and deterioration in the technical economic indices.

Mean monthly operating parameters for one of the blast furnaces are set out below:

Quantity of blast, m <sup>3</sup> /min	2600
Blast temperature, °C	798
Blast Furnace gas pressure, atm	0.64
Humidity of the blast, g/m <sup>3</sup> .	33
Ore charge	2.65
Hot blast pressure, atm.	1.78
System of charging	Three charges COOCCx
	Two charges OOC CGx

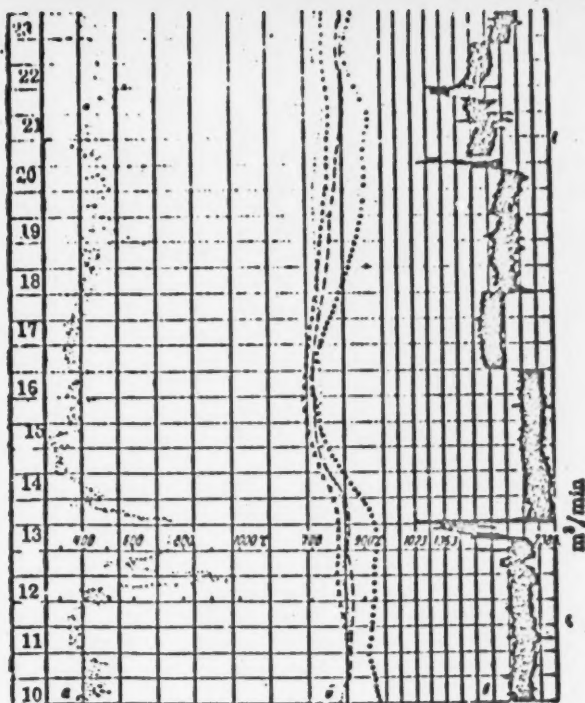


Fig. 4. Recorder charts for (a) blast furnace gas temperature (b) periphery gas temperature, and (c) air rate.

As the result of some displacement of the segments of the throat protection, formation of the slag column in the throat was unstable. The negative influence of this instability, however, could be completely eliminated by selecting the appropriate set of conditions and by the controlling action of the foremen. However, all the foremen of the same blast furnace were not of the same opinion as to the operating parameters and frequently applied opposing measures for regulating the furnace operation, and the gas flow was therefore unstable and slag break-outs and sets frequently occurred.

During the course of a month the furnace charging system was varied twenty-nine times (three times on account of changing the screen for sifting the coke fines). One foreman changed the charging system for the purpose of unloading the periphery, and his follow-on foreman for the purpose of loading up the periphery. The range of variation was from the COOCCx charging system to the system of three OOCx charges and two COOCCx charges.

Neither was there any single concept as to the quantity of the charge; one of the foremen considered that the charge should be 15 t (for the ore section) and another increased the charge to 16.6 t. During this period the blast varied from 2,500 to 2,700 m<sup>3</sup>/min.

Frequent changes in the system of operation of the furnace without any real foundation led to disturbances in the operation (break-through of the charge and setting). During the disturbances the hearth of the blast furnace was blocked, aided also by the high level of humidity of the blast (23-25 g/m<sup>3</sup>) with relatively low temperature of the hot blast (796°).

A discussion of the work of the foremen of this blast furnace by blast furnace foremen resulted in a single technological system being worked out, obligatory for all three foremen.

Measures were first taken to intensify the activity of the furnace by charging large portions of hearth clinker and partially manganese ore. The basicity of the slag was temporarily reduced, with some increase in

physical heating of the pig-iron. After a new drainage had been set up in the hearth and intense operation of the center of the furnace had been obtained, the foremen gradually established a higher melting rate. So that the furnace would be taking more blast, the number 2 tuyere was opened, the elbow of number 3 tuyere was cleared out and the charging system COOCx was temporarily applied. The blast was increased to 2700 m<sup>3</sup>/min. The temperature of the blast was then raised from 790° to 900°, at the same time raising the basicity of the slag. The humidity of the blast was reduced to 17 g/m<sup>3</sup>. Under these conditions the operation of the furnace was improved, and the furnace started to melt 230 t more agglomerate than under the disturbed conditions.

## CHANGING THE DOUBLE STUFFING BOX PACKING

E. A. Storozhik

(Foreman Fitter At The "Zaporozhstal" Blast Furnace Plant).

Frequent cases of failure of the stuffing box packing on the rotating charge distributor were encountered when operating with blast-furnace gas at high pressure.

If the stuffing box leaks it is necessary to replace it in order to avoid deterioration of the hopper or the rotating distributor. In addition, large gas leaks make it impossible to fill the hopper space with gas before lowering the large bell.

At the Zaporozhstal plant the stuffing box packing is changed without stopping the furnace. The work schedule is set out in Fig. 1.

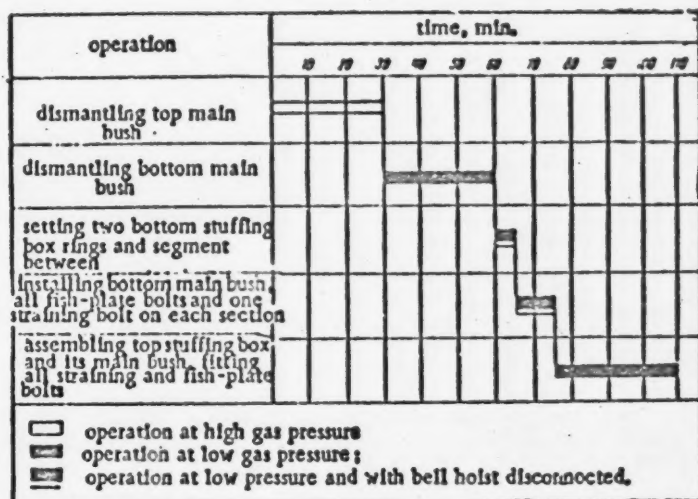


Fig. 1. Work chart of changing stuffing box

Each of the main bushes of the stuffing box (Fig. 2) consists of six segments joined together with bolts. The bottom stuffing box consists of two rows of 50x50 mm gasket with a spacer for lubricating. The top stuffing box consists of three rows of 32 x 32 mm gasket.

On the day scheduled for changing the packing, five pieces of gasket of the necessary length, impregnated with grease, are taken to the top of the stack together with small pointed rods and crooks (Fig. 3) for dismantling and assembling the main bush, bolts with well run-in nuts, keys (six sets for each size of nut), a knife for cutting the gasket, an oxy-gasoline set for cutting off bolts with damaged threads and a hammer and chisel.

Before changing the packing it is necessary to remove all grease from the main bush of the stuffing box and to take off all the oil pipes.

The team of fitters, changing the stuffing box (8-10 men), must be thoroughly familiar with the dismantling and assembly sequence of the stuffing box, the position of the joints, and so on, in order to avoid confusion during the work.

The segments of the top main bushing are taken out and dismantled during full operation of the furnace, with disconnected distributor, on basic working routine of the equalizing valves. In the case of severe deterioration of the large bell and pot, the charge is loaded onto the large bell with the small bell left open in order to provide better working conditions for the fitters.

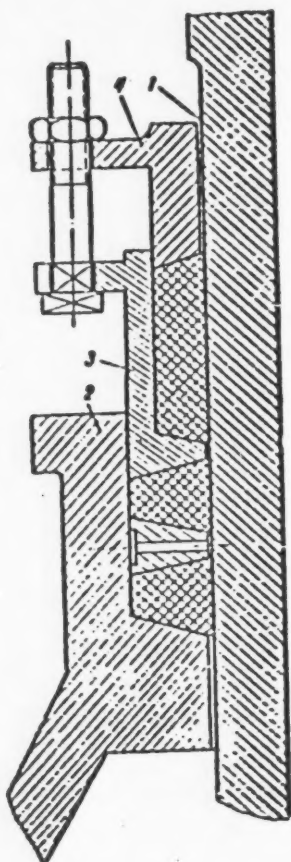


Fig. 2. Design of stuffing-box packing.

- 1) revolving funnel; 2) distributor body; 3) bottom main bush; 4) top main bush.

After all the main bushes have been removed, the groove on the stuffing box is blown through with gas while operating the large bell. Both main bushes must be cleaned of old grease; all the openings in the grease segment must be cleaned out.

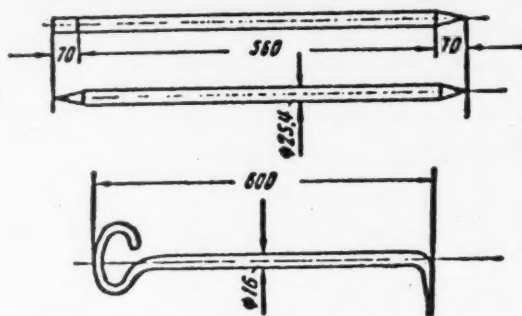


Fig. 3. Pointed rod and crook for changing stuffing-box packing.

The bolts are first removed (both jointing and housing) together with the six segments of the top main bush, and then, with reduced gas pressure, open exhaust valves and closed filling valves, the six segments of the bottom main bush are dismantled.

During operation of the large bell (and commencement of lifting the probes and commencement of closing of the small bell) the workers leave the platform arranged around the distributor, until blowing and steaming through the bush has ceased, after which dismantling is continued. If the atmosphere becomes too toxic the work is carried out in oxygen masks under the observation of a representative of the anti-gas unit.

The old stuffing box gasket is usually blown off after removing all the main bushes during operation of the large bell at low gas pressure. If the gasket does not blow off, it is necessary to raise the gas pressure to 0.3-0.4 atm.

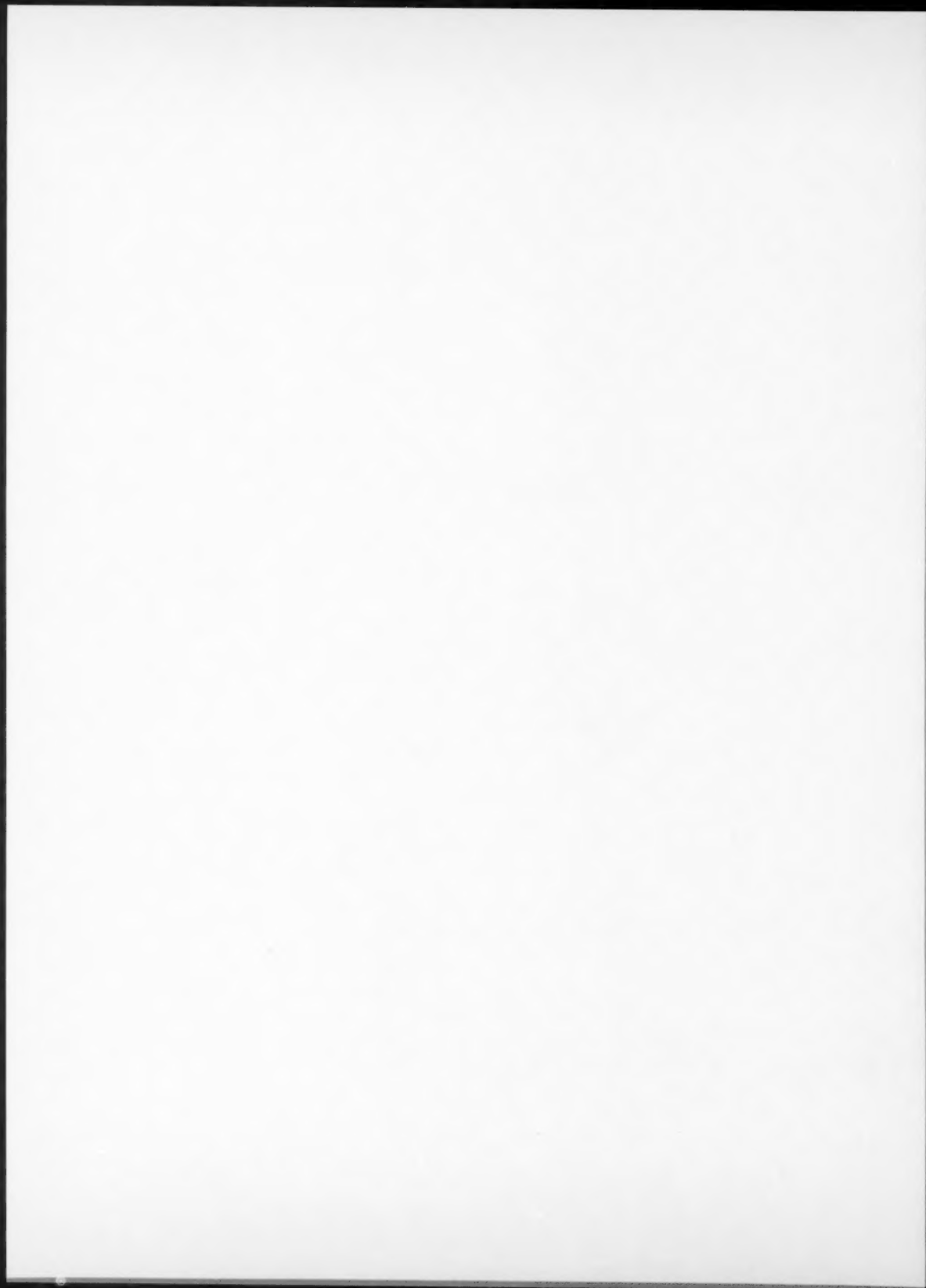
The bottom main bushes, stuffing box and segments are dismantled in a similar way to the upper, for applying grease.

Assembly of the stuffing box is started after finishing charging the furnace. After closing the large bell and opening the small bell the bell hoist is disconnected and assembly of the stuffing box is started. The first ring of the stuffing box gasket is fitted and the grease segments are assembled. It is necessary to check carefully the fit of the segments, so that the edge of one segment does not lie over the edge of the next. The second gasket ring is then fitted. The joints must be spread out, and twisting of the gasket avoided. The gasket of the second stuffing box is fitted at the same time as the assembly of the bottom row of main bushes is started. At first the main bushes must be set up against the reduction gear of the distributor drive (since this is the most tedious operation in assembling), following with the remainder. During this time the charge is built up on the large bell.

After assembly of the lower row of main bushes the bolts are tightened (one strain pin on each segment and all fish-plates). Only after tightening the bottom row of main bushes is permission given for lowering the large bell, and charging of the furnace commences with the charge distributor open.

Fitting of the top gasket rings and assembly of the top main bushings is carried out while the furnace is charged at normal pressure through the lowered small bell.

After completing assembly of the stuffing box it is necessary to connect up the grease fittings, connect in the rotating distributor and to grease the stuffing box continuously for one to two shifts. After this the pressure under the bell can be raised.



## CHARGING SYSTEM ON A HIGH-CAPACITY BLAST FURNACE

P. S. Balevich

S. M. Kirov Makeevka plant

The construction of high-capacity blast furnaces with a volume of 2000 m<sup>3</sup> requires a basic change in the charging system as the result of the appreciable increase in the quantity of raw materials charged into the furnace.

In publishing the present article for the purpose of discussion of the system of charging a high-capacity blast furnace, with a number of proposals which will require confirmation, the Editorial Board wishes to draw a wide circle of steel workers and metallurgists into the discussion. Both steel workers and scientific research institutes will doubtless bring forward a number of valuable proposals aimed at developing the most efficient charging system for a high-capacity blast furnace.

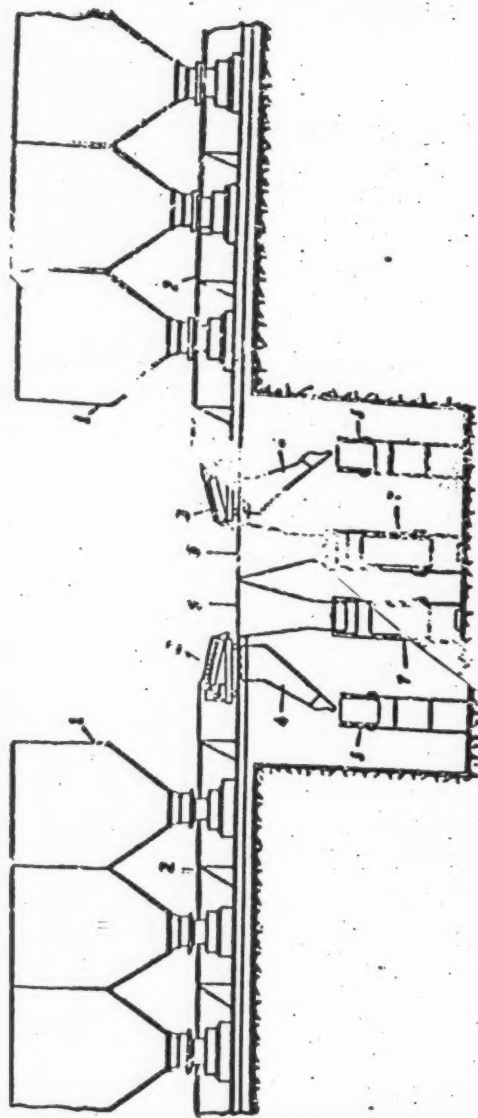
With present-day coefficients of utilization of effective volume, a blast furnace with a volume of 2000 m<sup>3</sup> will produce more than 3000 tons of pig-iron. The extraction of this amount of pig-iron by a blast furnace operating under the conditions, for example, of the S. M. Kirov Makeevka plant, requires an increase in the ore component of the charge up to 7600 tons without metal additions. The system of charging blast furnaces by means of the existing scale cars does not provide simultaneous delivery of this amount of raw materials into the furnace. If the present system of charging is to be maintained the capacity of the scale cars would have to be considerably increased.

It should be observed, however, that charging of the blast furnace by means of scale cars has several disadvantages. Each blast furnace carries a scale car maintenance team, comprising fitters, electricians and mechanics for repairing and regulating the weighing mechanisms. In order to provide normal operation of the scale cars, shift mechanic works on each shift in addition to the engineer and his assistant. On account of shortcomings in the construction of the valves of the ore bunkers, which do not provide the necessary degree of tightness, teams operate around the clock in the plants clearing the space under the bunker.

When the blast furnace is operating with hot agglomerate the working conditions of the scale car engineers are adversely affected. As the result of this, systematic clearing of material from the bunkers is not, as a rule, observed and there is additional accumulation of charging materials under the bunkers.

The conversion of blast furnaces to operation at higher gas pressure in the throat has considerably reduced the carry-over of blast-furnace dust. Of the charging material fed into the blast furnace, principally fine particles of 0.3 mm size are carried off with the blast-furnace gas. The presence of these particles in the charge is inevitable with the existing system of charging, since the raw materials, with the exception of the coke, are not screened before charging into the blast furnace. The best fluxed agglomerate contains up to 6% of fractions of a size 0-3 mm. The carry-over of blast-furnace dust promotes the early wear of the charging structure and the accumulation of dirt on the blast-furnace site, the factory and working area.

In order to obtain uninterrupted charging of large capacity blast furnaces with a volume of 2000 m<sup>3</sup> and to obviate the disadvantages connected with charging by means of scale cars, it is necessary to change the existing system of charging for a system employing a belt conveyor with bunkers equipped with rotating feeders.



Blast furnace conveyor-belt charging system.

According to this system of charging (see illustration) the bunkers for the ore part of the charge are arranged in one line. In this way it is possible without increasing the area under the bunker trestle to increase the capacity of the bunker to 200 m<sup>3</sup>. Since the capacity of all the bunkers must equal three times the volume of the blast furnace and for a blast furnace volume of 2000 m<sup>3</sup> the bunker capacity must be 6000 m<sup>3</sup>, the proposed system envisages 30 reinforced concrete bunkers, arranged in 15 units on each side of the skip pit.

This number and arrangement of bunkers provides a 100% reserve blast-furnace charge together with the facility for maintaining the conveyor-belt system.

The charge materials are fed into the receiving bunkers 1, and by means of the rotating feeders, according to the charge calculation, are fed on to the belt conveyor 2. The conveyor belt delivers the charge materials to the screen 3. The fine fractions of 0-3 mm are separated from the charge on the rotating screen. The screened part of the charge can be discharged direct to the stockyard from the fines bunkers 4 by means of the small skip-hoist 5.

The charge materials, after losing the fine fraction of 0-3 mm on the screen 3, are fed into the weighing hopper 6 for checking the weight of the charge, and then fed into the blast furnace by the skip 7 of the main hoist. All the materials are delivered simultaneously from all the bunkers in accordance with the calculated charge. It is thus possible, in addition, to neutralize the materials under the bunkers. It is possible by means of the rotating feeders to treat a delivery of materials from 0.2 to 12 kg. The materials are weighed on automatic scales at the treatment bench.

With loading of the charge at 150 kg per 1 m of conveyor belt, in one turn of the skip (60 seconds) the speed of the conveyor being 85 m/min (Kirov agglomerate plant) 12 tons of material can be taken in one ore-weighing hopper, and 24 tons in the two. In the course of an hour 1440 tons can be taken, which makes it possible to drive a high-capacity blast furnace to any required degree.

For maintaining this blast-furnace charging system one charge man is required for releasing the charge from the bunkers and one electrician at the charging station per shift.

The movement of the conveyor is interlocked with the operation of the skips of the main hoist.

The conveyor must automatically trip out when the ore-weighing hopper is filled with the necessary quantity of ore.

Charging of the furnace and operation of the charge distributor can be controlled from one control desk at the "charging" station.

The absence of fines in the charge materials makes possible a considerable increase in the driving of the blast furnace.

In order to obviate dustiness in the area under the bunkers and to provide normal conditions of work for treatment of the charge, it is necessary to set up a spray on each treatment bench for wetting the materials as they are delivered and to cover the separating screen for the ore section of the charge with casings and to install a suction fan.

The system for delivering the coke remains unaltered. The rotating screen must be enlarged in such a way that 3000 kg of coke is taken from the bunker to the coke-weighing hopper in the course of 60 seconds.

The power of the main hoist must also be increased so that the skip volume for coke is 9 m<sup>3</sup>.

In this way the employment of the blast-furnace charging system using a belt conveyor provides, fully automatic charging and a large power margin, utilization of charge materials under the bunkers, greater stability of the bell structure (absence of blast furnace dust), and more intense driving of the blast furnace.

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## OPERATING REGIME OF THE BLAST-FURNACE HOPPER SPACE

Engineer L. A. Lepikhin

### Magnitogorsk Metallurgical Combine

The operating conditions of blast-furnace bell structures have undergone a rapid change in recent years. The ore part of the charge loaded on to the bell has risen appreciably; up to 90% agglomerate goes into the charge, being more concentrated than ore, and reacts on the bell structure; the gas pressure has been raised with a consequent increase in the eroding action on all the components of the bell. The explosion probability in the hopper space has risen.

All this requires not only careful checking and supervision of the equipment but also the establishment of an operating regime for the bell structure which will provide normal operation over a long period.

At the Magnitogorsk Metallurgical Combine blast furnaces the present work program of the charging equipment on the hopper provides continuous delivery of 1.3-1.5 tons of steam per hour. Before opening the small bell the outlet valve is opened, closing at the moment when the small bell is  $\frac{1}{3}$  open. The pressure in the hopper before opening the exhaust valve and before filling reaches 0.3 atm. 20-40 seconds intervene between closing the exhaust valve and opening the filling valve with a normal charge. During this time 10-20 kg of steam is delivered to the hopper.

The operation of the hopper was investigated at No. 5 blast furnace with systems of charging OOCGx and COOCx. Tubes were lowered to a depth of 1 m in the hopper for gas sampling and for taking a thermocouple. 29 gas samples were taken during the investigation (Table 1).

The data of Table 1 show that air accumulates in the hopper during the charging-up period, and the oxygen content rises to 13-16%, which amounts to 75% when converted to air. A higher oxygen content is observed during the ore charge. In filling the hopper with gas, by reason of rarefaction, the oxygen content drops to 2-7%.

Table 2 gives data representing the change in pressure in the hopper of the blast furnace with different systems of charging.

After exhausting the gas before lowering the small bell the pressure falls to 0 and then rises to 90-100 mm H<sub>2</sub>O with the introduction of steam, and on lowering the small bell falls below 0 by 15-60 mm H<sub>2</sub>O. After closing the exhaust valve the pressure again rises; at this moment dust is ejected. A smaller vacuum (15 mm H<sub>2</sub>O) is observed with the coke charge and a greater vacuum-up to 60 mm H<sub>2</sub>O with the ore charge. On charging the agglomerate an average vacuum is obtained. The greater vacuum on charging with washed ore is explained by the fact that the washed ore produces a sharp drop in temperature in the hopper. The temperature in the hopper space, according to the thermocouple readings, is 200° C with a small drop to 180-190° with the ore charge, and also during the charging period. The small temperature change is explained by the thermal inertia of the thermocouple.

Hence, air enters the hopper space both as a result of the vacuum and through the layer of material. Analyses indicate that an explosive mixture is possible. The steam neutralizes the mixture.

TABLE 1

Gas Composition in the Hopper (Dry gas analysis, %)

After exhausting gas before lowering small bell					After lowering small bell				
CO <sub>2</sub>	CO	CH <sub>4</sub>	H <sub>2</sub>	O <sub>2</sub>	CO <sub>2</sub>	CO	CH <sub>4</sub>	H <sub>2</sub>	O <sub>2</sub>

## System COCCx with hot agglomerate.

14.8	27.2	0.3	1.0	1.2	Before filling with gas				
15.5	26.8	0.2	1.3	1.0	10.0	23.0	0.1	0.6	4.0
18.8	25.6	0.2	1.4	6.7	10.2	27.0	0.3	0.6	5.3
11.0	28.2	0.2	1.0	6.0	7.0	40.0	0.3	4.0	5.1
					After filling with gas				
					10.5	—	—	—	7.3

## System COCCx with washed ore

11.3	36.7	0.5	0.7	2.9	Before filling with gas				
11.3	20.7	0.4	0.4	4.7	4.7	4.7	0.6	—	15.3
5.7	39.7	0.4	—	13.7	6.3	10.9	0.1	0.4	13.0
5.6	10.7	0.1	0.3	12.4	3.7	7.3	0.1	0.3	16.3
					After filling with gas				
					14.4	24.2	0.1	1.0	1.2

## System COCCx with hot agglomerate

8.8	13.5	0.1	1.1	10.4	Before filling with gas				
4.6	8.8	0.1	0.6	15.0	10.0	17.9	0.2	0.6	7.1
—	—	—	—	—	3.3	7.1	0.1	0.4	160.0
3.5	7.9	0.1	0.4	16.2	After filling with gas				
					12.0	11.4	0.2	0.3	4.0

As was shown above, a gas mixture may occur (particularly with agglomerate) having a high CO content (up to 40%). The cause of the high CO content is not clear. In these cases the explosive range of the mixture is extended. A reduced ignition temperature is promoted by the presence of dust.

If an explosive mixture is formed while building up the charge, on lowering the agglomerate it may ignite and simple combustion then occurs.

If there is excess air in the hopper after the charge has been built up, then an explosion may occur on opening the filling valve. The source of ignition (the hot agglomerate) is at the bottom.

For the purpose of reducing the steam consumption the following measures can be recommended.

1. Restrict the supply of steam to the hopper, during charging up, to 0.5 t/hr increasing it before opening the filling valve to 1.5 t/hr. In addition, cut off the steam after closing the filling valve. Before opening the filling valve the hopper may be purged with steam through the open exhaust valve.

2. While building up the charge, the exhaust valve may be left open and no steam be supplied to the hopper. Before opening the filling valve the steam valve is opened. After closing the exhaust valve the filling valve is opened and gas is delivered to the hopper together with steam (this provided favorable results on two blast furnaces).

3. Before operating the filling valve, open the exhaust valve to draw off the air by reason of the temperature difference.

4. Before operating the large bell, purge the hopper with gas with open exhaust and filling valves.

5. Fill the hopper with nitrogen from the oxygen plant.

6. Purge the hopper with smoke from the air heaters, first filling with gas.

**TABLE 2**

Hopper Pressure, mm H<sub>2</sub>O.

After exhaust ing the gas	Before lowering the small bell	After lowering the small bell	During filling	Vacuum after lowering the small bell
System COCCx with hot agglomerate				
0	3260 3260 3130	100-0 100-0 0	6800-7500	-20 -20
System COCCx with washed ore				
0	2310 1770 1400	0 0 0	6800-7350	-60 -30 -15 -20
System COCCx with hot agglomerate				
0	2700 3500 3800	0 0 0	6800-7500	No vari- ation

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## STEEL SMELTING

### IMPROVED MANUFACTURING TECHNOLOGY FOR FREE-CUTTING STEEL

I. I. Fomin, A. D. Zaitseva and P. P. Konshin.

(Hammer and Sickle Plant)

Grade A12 free-cutting steel has been smelted for several years in the Hammer and Sickle plant in 75-ton open-hearth furnaces. The steel is poured on to 12-position bottom plates into 800 kg ingots. The chemical composition of the steel, in accordance with the standard specification GOST\*V 1414-54 is the following, in %:

C	Mn	Si	S	P
0.08-0.16	0.60-0.90	0.15-0.35	0.08-0.20	0.08-0.15

The high sulfur content in the finished metal, for the purpose of improving the workability of the steel on the cutting machines, adversely affects the plastic properties of the metal, particularly at high temperature, giving rise to what is called "red brittleness"; transverse cracks may occur on the ingots during crystallization; in rolling, the ingots are not held by the rolls well, slip occurs and hollows and seams appear.

The existing steel-smelting technology employs a phosphorus ferro-manganese alloy charged into the furnace with approximately 0.09-0.12% C content in the bath. Calculation of the quantity of phosphorus ferro-manganese is carried out according to the phosphorus. Deficiency in Mn is made up in the form of manganese-silicon. Both the alloys are fed into the furnace simultaneously for greater uniformity of distribution and lower losses.

During 1954-1955, for the purpose of improving the quality of the cutting steel, a large-scale investigation was carried out on a number of alloys as a result of which a new technology was developed for smelting and rolling the grade A12 steel. It was established on the first series of experimental alloys that in order to improve the quality of grade A12 steel it is necessary:

- 1) to maintain strictly the optimum filling speed of all the moulds in pouring;
- 2) to introduce the sulfur into the ladle in a metal case in order to achieve a more regular distribution and to reduce the loss;
- 3) to maintain the Mn:S ratio in the metal not less than 7.5 in order to improve the rolling conditions;
- 4) to drive smelting of the metal and attain the greatest reduction of the metal.

In the second series of experimental smelts an investigation was made of the influence of supplementary preliminary reduction of the metal with blast-furnace ferrosilicon to the amount of 7 kg/t of steel. The ingots of these smelts, poured with optimum filling speed of the moulds up to the shrinkage head (over 3 minutes) did not display any cracks or fractures in rolling. The table shows that two factors are decisive for suitable manufacture: the degree of reduction of the bath before pouring the metal and filling rate of the moulds with the metal. With a dead bath and filling time of the moulds less than 3 minutes, cases of fracture of the slabs in rolling amounted to 22.6-31%. Under the same conditions, but with filling time of the moulds greater than 3 minutes, the cases of fracture were reduced to 1.9-0.23%.

\* GOST - All-Union State Standard - Publishers note.

Reduction of the metal.	State of the bath before pouring.	Filling time of the mould to the shrinkage head, minutes					
		Less than 3			Over 3		
		No. of Ingots rolled	No. of Ingots with fractures		No. of Ingots rolled	No. of Ingots with fractures	
			No.	%		No.	%
Without preliminary reduction.	Dead	443	100	22.6	160	3	1.9
	Bubbling	231	164	71.0	107	49	45.8
With preliminary reduction by blast furnace ferrosilicon	Dead	305	95	31.1	851	2	0.23

Bubbling of the bath has a particularly adverse effect on the quality of the metal; with rapid pouring of the metal into the moulds fracturing of the ingots amounted to 71%. Even with slow pouring (over 3 minutes) bubbling of the metal in the furnace yielded 45.8% cases of fracture during rolling.

Good results were obtained with preliminary reduction of the metal with blast furnace ferrosilicon and with slow pouring of the metal (filling rate of the moulds over 3 minutes). In this case fracturing of the ingots during rolling was practically absent (out of 851 rolled ingots fractures were observed only on two).

The results of the work indicated the possibility of completely obviating slipping and considerably reducing defects through cracks on the ingots and pockets on the end faces of the slab during rolling.

The main points in the new technical code for smelting and pouring grade A12 steel can be reduced to the following.

- 1) Grade A12 steel to be poured at a filling rate of the moulds up to the shrinkage head not less than 3 minutes, not allowing turning during this time.
- 2) Rimming to proceed actively with the greatest possible reduction of the metal. For this purpose the bath should be previously reduced with blast-furnace ferrosilicon and 7-10 kg/t of metal charge introduced, depending on the quantity of manganese-silicon introduced.
- 3) Not to allow melts with low carbon content for grade A12 steel melts.
- 4) To maintain the Mn:S ratio not less than 7.5.
- 5) To serve the sulfur into the ladle in a special metal case.

## THE USE OF MAGNESIUM IN CONVERTER STEEL MANUFACTURE

K. S. Prosvirin, V. I. Baptizmanzky  
(Dnepropetrovsk Metallurgical Institute)

and

M. P. Kuznetsov, V. D. Umnov  
(Dzerzhinsky Plant)

For the purpose of improving the plastic characteristics of Bessemer rails experiments were carried out for treating the steel with magnesium in the Dzerzhinsky Bessemer plant.

The introduction of metallic magnesium into the liquid steel is accompanied by a considerable luminous effect and the evolution of a large quantity of white smoke. After a number of preliminary experiments, the magnesium was introduced into the molten steel in the form of silico-magnesium alloy. Employing this method, the intense luminous effect and evolution of large quantities of white smoke were not observed.

In the treatment of the rail steel with magnesium, the 45% ferrosilicon introduced into the ladle during reduction was completely or partially substituted by a corresponding quantity of silico-magnesium alloy (64% Si, 11.8% Mg) in order to provide the required silicon content in the finished steel.

In five of the ten experimental smelts the ferrosilicon was completely substituted by silico-magnesium and in the remaining smelts only partially. The quantity of reducing agents added is given in table 1.

TABLE 1

Quantity of Additions to the Smelt, kg.  
(Weight of smelt 22.5 t)

Number of smelts	Ferro-Manganese	Ferro-silicon	Silico-magnesium	Aluminum
5	210	53	20	3.3
5	210	-	60	3.3

Ferro-manganese was introduced into the converter in the molten form after finishing blowing, the remaining reducers being added to the flow during pouring of the metal into the teeming ladle.

The technology of blowing, teeming the steel and rolling the ingots in the experimental smelts was the same as for ordinary smelts. In this way the experimental smelts differed from the ordinary process only by the fact that the ferro-silicon was completely or partially substituted by silico-magnesium.

Tables 2 and 3 show the brief characteristics and the results of check tests together with supplementary investigations of the quality of the rails from the experimental smelts.

TABLE 2

Brief characteristics of experimental smelts.

Smelts	Chemical composition of steel %						Standard scale grain size	
	C	Mn	Si	S	P	Ni	Actual	Natural
With the addition of 20 kg silicomagnesium	0.53	0.74	0.19	0.037	0.056	0.015	4	4-3
	0.52	0.71	0.19	0.038	0.057	0.016	4-3	3-2
	0.51	0.82	0.20	0.038	0.057	0.017	6	3-2
	0.59	0.68	0.18	0.037	0.061	0.017	3	3-2
	0.52	0.71	0.19	0.038	0.060	0.016	4-3	4-3
With the addition of 60 kg silicomagnesium	0.51	0.84	0.22	0.038	0.057	0.017	6	3-2
	0.52	0.76	0.23	0.035	0.056	0.018	3	3-2
	0.57	0.69	0.18	0.037	0.056	0.017	4-3	3-2
	0.50	0.81	0.23	0.036	0.057	0.021	4-3	3-2
	0.57	0.69	0.23	0.036	0.057	0.024	4-3	4-3

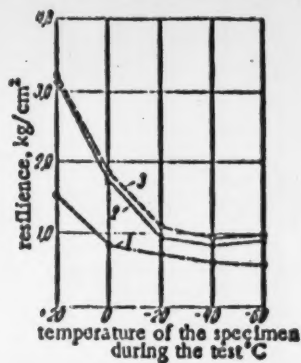
TABLE 3

Mean data of the mechanical tests.

Quantity of alloy added, kg.	Creep limit kg/mm <sup>2</sup>	Ultimate strength kg/mm <sup>2</sup>	Relative elongation, %	Relative compression, %	Brinell hardness	Standard scale grain size	
						Actual	Natural
20	43.5	81.0	12.3	27.3	246	6-3	4-2
60	44.6	84.0	12.3	27.6	250	6-3	4-2
check smelts	46.3	85.5	10.6	21.0	260	1-0	2-1
Normal production smelts	45.0	83.5	10.5	19.5	248	2-1	2-1

The data set out in Table 3 and Fig. 1 show that the test rails have good mechanical characteristics and completely meet the specified requirements. In respect to resilience, the rails of the experimental smelts approach the rails rolled from open-hearth steel. The influence of magnesium additions on the grain size is associated probably with the variation in the surface tension of the metal and supplementary formation of centers of crystallization.

The magnesium also has a certain influence on the quantity and nature of the nonmetallic inclusions. The nonmetallic inclusions in the metal treated with silico-magnesium are finer and in smaller quantity than in steel from normal smelts. Samples of metal from five of the experimental smelts and from six smelts not employing magnesium were investigated by the electrolytic method for metallic inclusions. The mean content of nonmetallic inclusions in the steel from the experimental smelts was 0.0263% and in the smelts not using magnesium 0.0427%.



Relationship of resilience to temperature of the specimen at the time of the test and the amount of additions of silico-magnesium alloy;  
 1) Check smelts; 2) Smelts with 20 kg silico-magnesium addition;  
 3) Smelts with 60 kg silico-magnesium addition

It should be added that on introducing magnesium into the steel, some reduction of the sulfur content is observed. For example, the mean sulfur content in the metal of the experimental smelts differed by 0.002-0.006% as compared with the mean sulfur content in the finished metal.

Hence, the experimental smelts show that with introduction of magnesium into the molten metal the grain is reduced, there is a reduction in the quantity of nonmetallic inclusions and in the sulfur content, and there is an improvement in the plastic characteristics of the rails.

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## IMPROVING THE DURABILITY OF FURNACES

Ya. G. Privezentsev  
(No. 1 Open-hearth furnace Manager)

V. Ya. Ignatenko  
(Brick-work foreman on the Illich Zhdanov plant)

The thermal capacity and operation of an open-hearth furnace depend substantially on the efficiency of the regenerators. Inadequate stability of the Dinas checker of the regenerators is the main obstacle in increasing the smelting rate.

Gases at a temperature of 1600-1650° carry off dust consisting of iron oxides from the work area; these oxides are deposited on the surface of the acid brick of the checker and form a compound with a fusion temperature of 1150°. The brick rapidly disintegrates and melts, becomes thinner and weaker, while the checker loses its structural strength and the upper rows collapse. The channels of the lower rows become crusted over.

The basic iron dust of the stack gas destroys the acid Dinas brick, but does not react with materials having basic or neutral chemical characteristics. One of the most important requirements for the brick of regenerator checker is a high slag stability. For this reason our factory began to employ slag-resisting magnesite chrome brick for the upper rows of the checker, starting in 1952. At first, use was made of magnesite chrome brick available at the factory and manufactured at the Panteleimonov plant with the following composition 27.0%  $\text{Cr}_2\text{O}_3$ , 43.0%  $\text{MgO}$ . The top four rows were laid from this type of brick, continuing with Dinas checker in the lower rows.

After two runs on the furnace top (460-500 smelts) the verticle surfaces of the brick checker (top two rows) were covered with a layer of crystals to a thickness of 8-10 mm and the linear dimensions of the channels were reduced by approximately 10%. The third and fourth rows of magnesite chrome brick were in a better condition and displayed a negligible layer of crystals. The lower rows of Dinas brick, after the second run of checker operation, were fused to such a degree that it was necessary to renew them completely.

Later on, for the purpose of increasing the stability of the regenerator chambers and reducing furnace stoppages for cold repairs, the Dinas brick in the checker was replaced by Chasov Yar refractory brick and more recently with refractory brick of our own manufacture having the following composition: 35%  $\text{Al}_2\text{O}_3$ , 62.0%  $\text{SiO}_2$ , 2.0%  $\text{FeO}$ , and 0.5%  $\text{CaO}$ . The life of the checker built from chrome magnesite and the refractory brick amounted to 826 smelts during 1955.

At the present time at the cold overhaul period the checker is cleaned with a ramrod and blasted with compressed air (3-4 atm). This improves the operation of the regenerators. After two runs the top three rows of magnesite chrome checker are renewed while the refractory brick is left for further operation.

Furnace stoppages for hot repairs to the front rows amounted to 100 furnace hours per month. Usually after 120-150 smelts the furnace was stopped for hot repairs to the roofs and columns of the front rows. The cost of each such hot overhaul amounted to as much as 18-20 thousand rubles, the furnace program of the plant was interrupted, the fuel consumption was increased, and so on.

In order to obviate furnace stoppages for hot repairs to the front rows, the roofs and columns were built from ordinary magnesite chrome brick with metal cladding. The strength of these elements was raised to the strength of the furnace roof. The economic effect of this amounted to more than 400 thousand rubles.

On increasing the durability of the front rows the necessity occurred for increasing the durability of the working space roofs. The furnace had frequently to be put on to cold overhaul on account of the tendency to wear at the back wall and the base brick along the front and rear walls after only 220-230 smelts.

For building the rear wall (above the clinker zone) use was made of magnesite chrome brick with metal cladding, while instead of the Dinas base brick for the straight roof along the front and rear walls, abutments were set up out of dressed magnesite chrome brick. As a result of this the durability of the roofs was increased to 270-310 smelts.

These measures promoted an appreciable increase in steel smelting, a saving in fuel and refractories, and a reduction in the cost of the metal.

## THE CHOICE OF REFRACTORIES FOR REGENERATOR CHECKERS

Engineers S. L. Gribov and A. M. Banschikov  
(Kirov Machine Construction and Metallurgical Plant)

When basic materials are used for the roof, the length of the run and the productivity of open-hearth furnaces at several plants are governed not by the stability of the main arch, but by the stability of the regenerator checker. The Dinas brick commonly employed for building the top rows of the checker quickly melts at the high temperatures at the top of the checker, giving rise to interruption in the furnace performance and early stoppage for overhaul.

For normal operation of the Dinas checker the maximum temperature of the top of the checker should not exceed 1250°, which cannot be maintained with intense driving of the furnace with a magnesite chrome roof.

The open-hearth furnaces of the Kirov works are fired with oil and operate on the scrap process. The volume of the checker is 104 m<sup>3</sup> each way. The checker is laid zig-zag, with cell dimensions 150x120 mm. The ratio of the checker to the area of the hearth is 3.43 m<sup>3</sup>/m<sup>2</sup>, and to the weight of the melt 1.4 m<sup>3</sup>/t. The maximum thermal loading is 16 million cal/hour.

The furnaces are fitted with Moshkarev atomizers with air atomization at an atomizer pressure of 4.5-8 atm. Secondary air is delivered from a blower. The automatic regulators of the thermal regime are set according to the individual parameters. The valves are operated in accordance with the temperature of the checker top.

For a long time the regenerator checker was built up according to the usual system: 2/3 of the height from the bottom of ordinary refractory and the top third of Dinas brick. The normal durability of this type of checker corresponded to two runs of the Dinas roof—500-550 smelts. After 220-270 smelts, simultaneously with the overhaul of the main roof, the checker was partially overhauled. A typical feature of this was the fusing of the 4-6 top rows of Dinas brick in the middle section of the checker, nearer to the stone wall. The fused portions were dismantled, and after cleaning, the remaining sections of the cells were replaced by fresh Dinas brick; the volume of the replaced section sometimes comprised 25% of the total volume of the checker.

Work was carried out on an open-hearth furnace for the purpose of selecting resistant refractories for the regenerator checker on the main open-hearth furnaces. Four variants of construction were tried for the top rows of the checker.

Variant 1. Checker made up of standard class B fireclay brick (230x115x65 mm), with cell dimensions 120x120 mm had a life of 296 smelts. The temperature at the top of the checker did not exceed 1300° during the tests.

Fusing of the top rows was less than with Dinas checker. The main cause for repairs to the checker was the formation of slag incrustations in the middle row of the checker, as a result of which the furnace draft fell off sharply. The incrustations had a high mechanical strength and could not be removed by ordinary cleaning methods.

Variant 2. The fifteen top rows of the checker were built from checker brick (300x150x75 mm) with a cell dimension 150x120 mm. The Al<sub>2</sub>O<sub>3</sub> content in the checker brick was in the range 40-42%. The mean temperature at the top of the checker was 1350-1380° (maximum 1450°). The life of the checker was 290 smelts.

Before overhaul, the strength of the lining and the pressure in the working space increased sharply and the thermal performance of the furnace fell off noticeably.

During overhaul intense fusing was observed on the eight top rows of brick together with congealing of the slag formed in the lower part of the lining. Incrustation was so intense as to form a solid block of slag and brick in the lower structure of the lining in places, and blasting had to be employed to remove it.

The thermal loadings and temperature regime of the checkers were therefore reduced, setting a maximum temperature of 1350° at the top of the checker. Under these conditions the life of the checker was increased to 600 and more smelts, while on overhauling, a considerable part of the brick of the bottom half of the checker could be again employed in the construction after cleaning.



Fig. 1. Appearance of the top row of chrome magnesite brick of the checker after 276 smelts.



Fig. 2. Nature of the deposit on the first row of the checker after 490 smelts (top six rows chrome magnesite brick).



Fig. 3. Appearance of the dismantled section of the checker from the first of the six rows (chrome magnesite brick) after 490 smelts.

Variant 3. For the purpose of preserving the Dinas brick from the chemical action of the foundry dust in the high temperature zone in the top rows of the checker, slag-resisting chrome magnesite brick was employed. In the usual checker, built up to two-thirds from standard fireclay and one third from Dinas brick with cell dimensions 130x130 mm, the four top rows were constructed of standard chrome magnesite brick with dimensions 230x115x65 mm. The mean temperature at the top of the checker was 1350-1360°.

After the 272nd smelt the draft in the furnace deteriorated appreciably. At the 279th smelt the furnace was stopped for overhaul.

On account of the intense fusion and consequent mechanical disintegration of the three top rows of Dinas, located immediately under the chrome magnesite, the checker was dismantled. The rows of Dinas underneath were also fused to a considerable extent. The fireclay brick maintained its dimensions and was in satisfactory condition. The chrome magnesite brick in the first row from the top was uniformly coated with a slag crust to a thickness of 10-15 mm, forming a solid skin which was difficult to separate from the brick. The chrome magnesite brick of the second row maintained its initial dimensions and mechanical strength. The surface of this brick was coated with a slightly sintering thin layer of dust which could be readily separated from the brick.

A similar type of checker was tested on a second furnace, but with a mean top temperature of 1300°. The life of this checker was increased up to 598 smelts. The nature of disintegration of the brick was the same as in the first case.

The checker tests showed that the employment of chrome magnesite brick in the 4-6 top rows arranged above the Dinas does not provide sufficient stability of the checker with intense driving of the thermal regime of the furnace; the main cause of the checker falling was disintegration of the Dinas brick.

Variant 4. The Dinas brick was replaced by a fireclay-brick checker more resistant to slag, having a 40-42%  $Al_2O_3$  content. The top six rows were made up of ordinary chrome magnesite brick, the next eight rows from fireclay checker brick, and the remainder, down to the bottom, of standard fireclay brick. The cell dimensions were 150x120 mm.

The furnace was operated with a Dinas arch on the working area with thermal loadings up to 16 million cal/hr. The mean checker top temperature was 1300° and the maximum 1350-1400°.

The checker was examined for the first time after 276 smelts (fig. 1). On this occasion there was no evidence of caving, shifting of the rows, of fusing of the brick. The horizontal surface of the top row was coated with slag having a crystal structure to a thickness of 10-15 mm, which was separated from the brick without particular difficulty. The vertical surfaces of the brick carried a slight deposit of dust and slag. After removing the outer slag crust and blowing the checker with air the cells were the original size. The checker was returned to service.

The checker was examined for the second time after 490 smelts. The slag layer on the horizontal surface and the slag and dust deposit on the lower rows of chrome magnesite brick were greater than at the first inspection (fig. 2). On removing the slag crust, partial disintegration of the first-row brick was observed, with spalling and degeneration of the structure. The chrome magnesite brick of the lower rows was in a satisfactory condition. Fig. 3 shows a dismantled section of the checker of the six rows. It is seen from fig. 2, and 3 that the main mass slag is formed on the first row of chrome magnesite brick and in the remaining rows the cells are fairly clean. The fireclay checker brick set up under the chrome magnesite was in good condition and completely maintained its dimensions and mechanical properties. The brick surface was coated with a thin layer of porous iron slag.

It should be noted that slag formation and wear of the checker at the points of more intense gas flow were less than in the colder parts of the checker.

Comparison of the results of checker tests on two similar type furnaces, the one having a regenerative checker made up of checker fireclay brick in the top section, and the other with a checker the top six rows of which were made up of chrome magnesite brick, indicates that with similar thermal loadings and similar initial temperature of the checker top, the main secondary air heating temperature in the checker with chrome magnesite protection was lower by 80°. This temperature reduction can be explained by the porous slag crust formation, reducing the coefficient of thermal conductivity of the brick in the top rows of the checker.

The reduction in the coefficient of thermal conductivity reduces the storage capacity of the brick by increasing the passive layer not participating in the heat transfer. Hence, chrome magnesite brick can be employed in regenerator checkers only in the smallest quantities, for protecting the basic active part of the checker of fireclay brick from the temperature and chemical action of the gaseous medium.

The following conclusions can be drawn on the basis of the tests:

1. In building Dinas brick into the top checker rows the thermal loadings of the furnace and driving of the smelt are restricted, since the maximum temperature at the top of the checker must not exceed 1280°; a Dinas brick checker cannot be employed when operating with basic roofs. Top protection of Dinas checkers with chrome magnesite brick is ineffective; the Dinas brick under the protective chrome magnesite rows is disintegrated.
2. Fireclay checker brick with a 40-42% alumina content displayed satisfactory stability with checker top temperature not exceeding 1350°.
3. Protection of the top of the fireclay checkers with chrome magnesite brick appreciably increases the stability of the checker. The employment of chrome magnesite brick only in quantities necessary for the protection of the lower fireclay brick can be recommended with basic roofs of the active area and maximum checker top temperature of 1450°. The top two to three rows of chrome magnesite brick were replaced by new brick and the cells were cleaned out with a ramrod and blown with compressed air.

On inspection of the checker after 699 smelts the top row was again cleaned, the external appearance indicating that it was in the same condition as at the first inspection, and the checker was returned to service.

## KRIVOI ROG SOUTHERN ORE CONCENTRATION PLANT

On the right bank of the river Ingulets lies the enormous site where day and night without cease there is heard the roar of drills and excavators, and the winning of iron ore goes on. Away on the hill are the enormous factory blocks and multi-storey housing quarters. This is the Krivoi Rog Southern Ore Concentration Plant.

Until recently, only rich ore not requiring concentration came from the Krivoi Rog iron ore-field. When low grade ore was encountered, it was dumped for lack of a concentration plant. The concentration of low-grade ores alongside the treatment of rich ores for blast-furnace smelting is one of the major problems of the Sixth Five-Year Plan.

Commencing construction in May 1953, the first stage of the Krivoi Rog Southern Ore Concentration Plant was commissioned in the second half of 1955, providing for the exploitation in the Krivoi basin of iron ores with a 35-37% iron content. There are very large reserves of this type of ore in the basin, several times exceeding the reserves of rich ore.

Iron quartzites are made up principally of the two minerals magnetite and quartz. The magnetite forms bands of ore to a thickness of 0.3 to 1.3 mm in the quartzite. The quantity of injurious additions in the ore is very small (phosphorus 0.06%, sulfur 0.05%).

The plant is located 10 km from Krivoi Rog, to the south-west of the Krivoi Rog Metallurgical Plant, with a tramway connection to the town. The plant comprises the quarry, concentration and agglomerate plants and a range of workshops.

The planned capacity of the plant is 5.25 million tons of agglomerate annually.

The Krivoi Rog Southern Ore Concentration Plant was planned by the Yuzhgiiprorud and Mekhanobr\* Institutes, incorporating all the latest achievements of Soviet and foreign science and engineering and the most advanced mining technology.

9 million tons of iron quartzite are delivered to the enrichment plant from the quarry every year. The quarry is equipped with power excavators and cable-tool drilling machines; transport at the quarry is electrified.

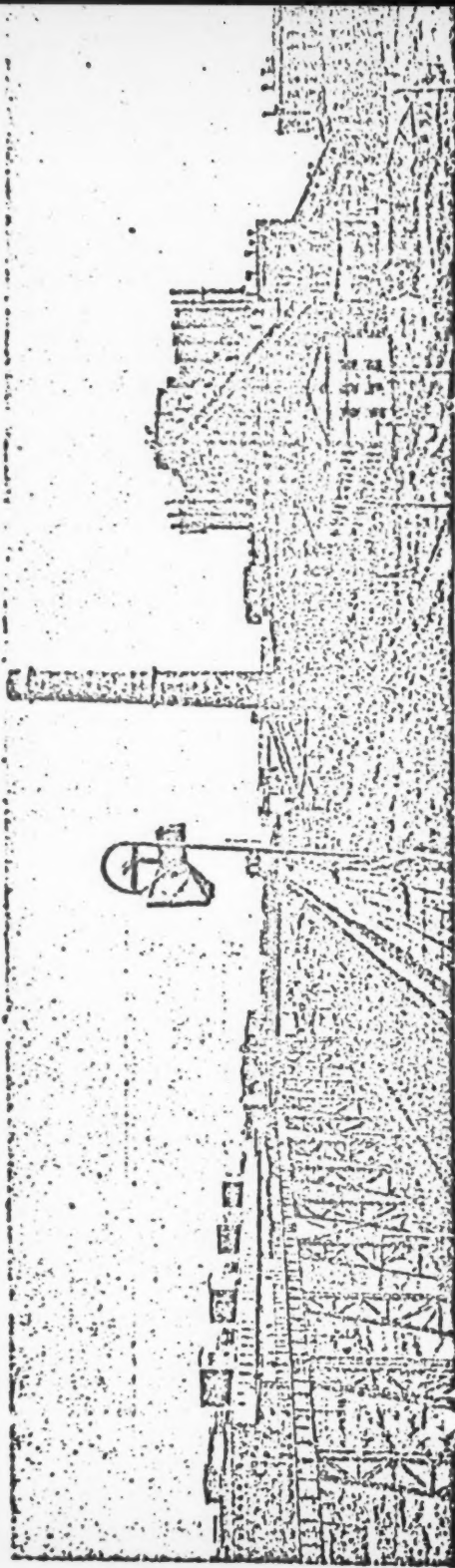
The ore is delivered to the factory from the quarry in lumps up to 1200 mm by electric trains in 60 ton dumping wagons and discharged into a power cone crusher with a capacity of 2000 tons of ore per hour. The primary ore is first crushed to 250 mm in three stages in the cone crushers before concentration. On account of the fine dissemination of the ore material, the ore is treated in two stages, grinding to 0.5 and 0.1 mm in ball mills. The ore is divided into concentrate and tailings on magnetic belt separators.

The total concentrate output with 60% iron content is 52% with 84.4% iron extraction. After dehydration in vacuum filter drums, the concentrate is taken off to the agglomerate factory for sintering. The tailings from the concentration plant are transported and dumped.

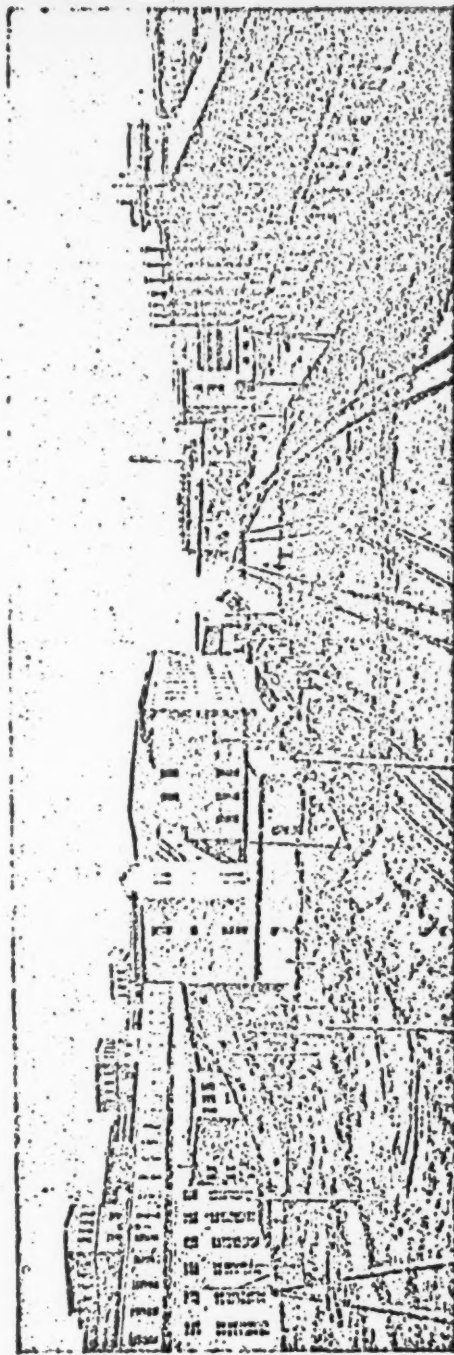
The power agglomerate factory has ten machines (two buildings of five agglomerate conveyors) each with a sintering area of 75 m<sup>2</sup>. Fine Krivoi Rog rich ore (10-0 mm) and limestone are added to the charge for intensifying the sintering process and fluxing the agglomerate. The agglomerate factory provides blast-furnace fluxing agglomerate with a 55% iron content which is despatched to the Southern Metallurgical Plants.

Large comfortable living quarters for the quarrymen and process workers are constructed alongside the plant.

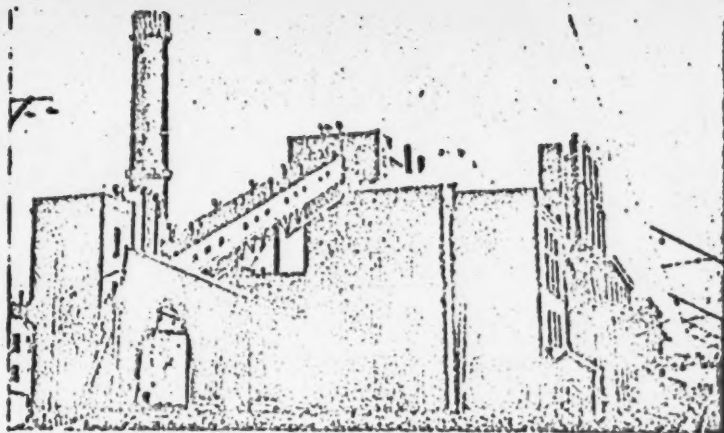
\* Mekhanobr = Scientific Research Institute for Mechanical Concentration of Minerals—Publisher's note.



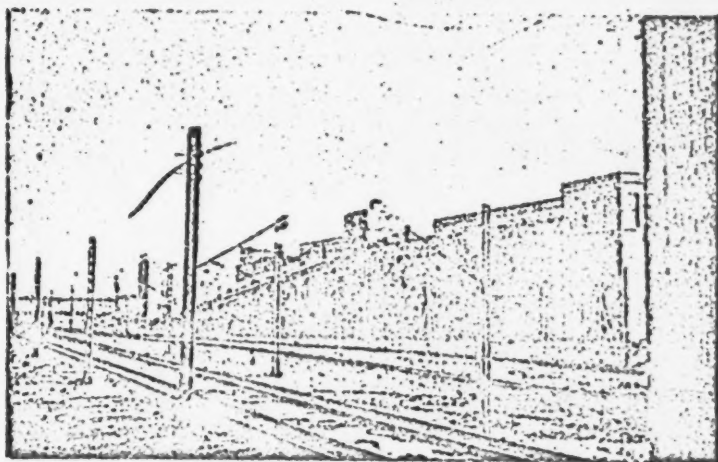
General view of the plant



General view of the plant

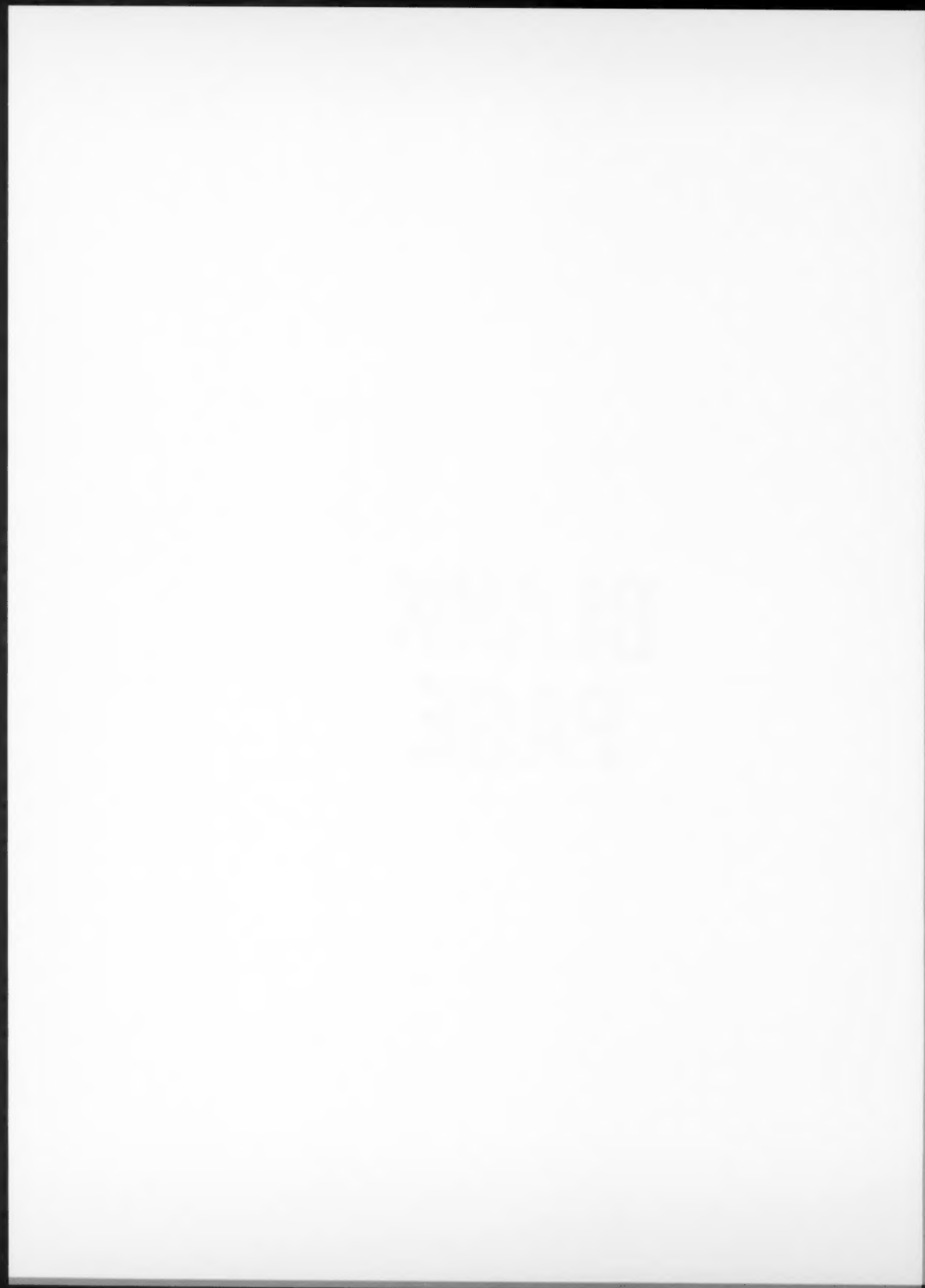


Main ore-concentration block.



Primary agglomerate mixing block.

On the basis of experience gained in the operation of the Krivoy Rog Southern Ore Concentration Plant it is intended to build several more such plants in the Krivoy Rog field.



## ROLLING MILL AND TUBE PRODUCTION

### EXPERIENCE ON COMMISSIONING THE 2800 PLATE MILL

D. I. Timofeyev

(Foreman, Voroshilov No.2 Sheet Mills)

The giant 2800 sheet mill started operation at the Voroshilov plant in November, 1955.

The mill can handle sheets of a thickness from 4 to 50 mm, width 1500 to 2600 mm and length up to 20 m from various grades of steel. The mill output should reach up to 1,100,000 tons annually.

The first stage of the mill takes the form of a continuous run consisting of four parallel lines arranged in three bays and interconnected by pullover transfers (see diagram).

The run consists successively of the feeding machines, heating furnaces, vertical stand, and two-high reversing stand with all the auxiliary equipment and hydraulic press equipment, and a four-high universal reversing stand with all its equipment. After the stands there are the roll-bed coolers, straightening machine for thin sheets, roll-stand coolers and machine for correcting thick sheets.

Then follow the tilter and twin inspection roll bed with roller transfer, marking-off machine, the first cross-cut shears with 600 t pressure, and tilting table, scrap rejector, conveyor for transporting the scrap and scrap shears.

The scrap is taken off to the scrap shears by conveyor by means of a duplex ejector and after shearing is ejected into the trough by the simplex ejector.

After the first shears the metal can be passed along one of three lines:

- 1) to the left to the stacker,
- 2) to the right to the thin sheet shears, or
- 3) to the right to the plate shears, and then to the stacker.

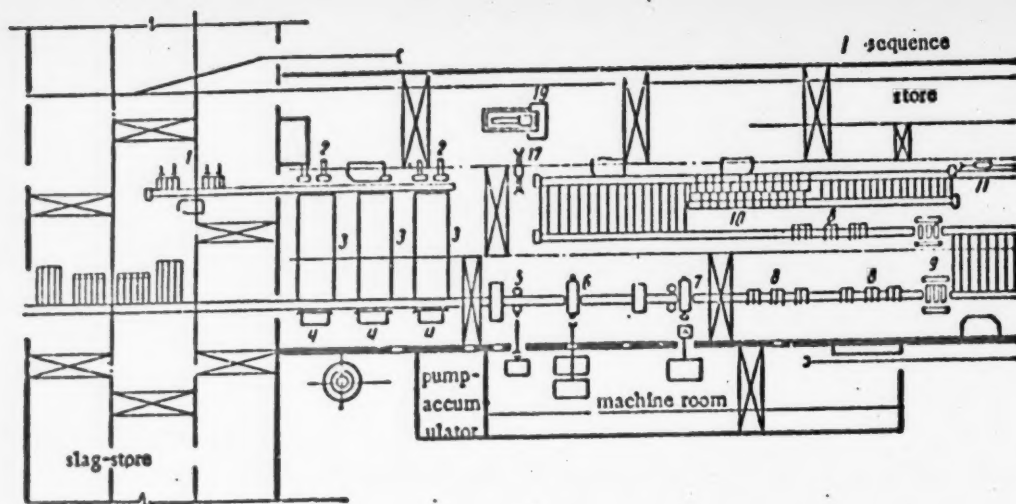
The second stage of the mill (thermal section) is not yet constructed. It will occupy three bays representing a continuation of the existing bays of the factory building.

The mill is automatically operated, having in addition to a large number of electric motors on different systems and powers, special hydraulic and pneumatic drives. For example, one four-high universal stand alone with manipulators and other auxiliaries carries seven electric motors and twenty-five hydraulic drives operating at a pressure of 80 and 200 atm.

In installing this type of mill it is obvious that serious difficulties had to be overcome, the more so since entire units and separate mechanisms had to be designed in planning the mill.

During the operation stage a number of changes in the design were gradually introduced and new machines were added not envisaged in the plan. The rolling technology was considerably modified.

One serious shortcoming of the plan was that all of the working rolls were manufactured from grade 60 XH forged steel. Conversion to ordinary iron rolls used on sheet mills without large-scale conversion of the stands themselves was impossible since the rolls did not possess the necessary strength.



Plant Layout for the 2800 mill:

- 1) Feed table with rammer; 2) furnace push-bars; 3) reheating furnaces; 4) furnace dampers; 5) straightening machine; 6) tilter; 7) marking-out machine; 8) drop-down cross cutting shears; 9) transfer car; 10) scrap shears; 11) drop-down cold shears with goose necks.

The plant management decided first to work on reducing the time expended in changing the rolls. After some improvements and modifications in the design of the machine fittings, the roll-changing time was reduced from 3-4 hours to 20-30 minutes. The rolls are now changed more frequently—each shift. However, even after these improvements, on account of the special nature of treatment of the soft steel rolls, a number of sections could be rolled for only a resulfured period. Important orders were not met and there was a high reject level. In addition, the expenditure in rolls increased to such an extent that the roll turners could not keep up the roll stock and, toward the middle of the year, nearly the whole of the roll stock intended for the year was expended. Therefore, simultaneously with commissioning the mill, a start was made at the plant for setting up an assembly for applying a hard alloy to the roll surface.

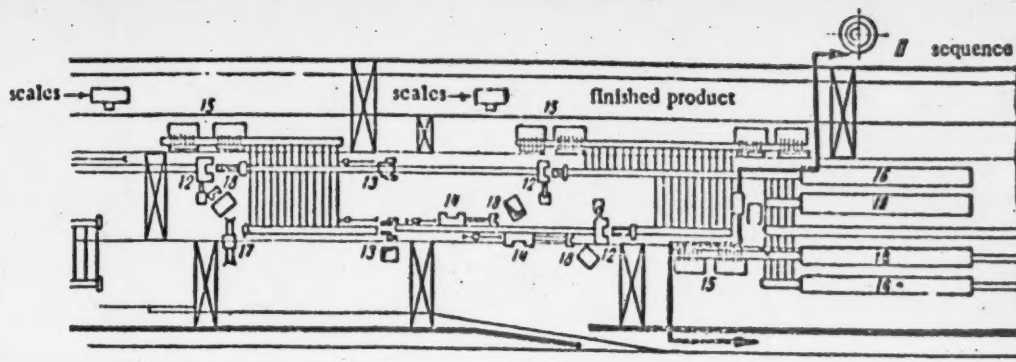
The life of the roll was extended to four times, although the technology of fusing-on the alloy still requires considerable improvement.

The plan for the mill envisaged that the sheets would come off perfectly uniformly and without surface defects. In practice, however, the sheets are reduced at the ends. This would not be particularly important if the mill were adapted for transporting such sheets along the whole of the pass.

In addition, the project did not envisage treatment of the metal in the run. The coolers and delivery frames along which the sheets slowly pass were not adapted for this.

All the tables, starting with those approaching the furnaces, were designed without inter-roll plates. Hence, the slabs fell downwards on the charging and receiving beds before and after the furnaces. On the beds after the mill and on both shear lines the sheets with reduced ends struck the rollers, tearing them from the foundation and fracturing the bed plates. The thin sheets, even in the case of perfectly uniform sheets, bent under their own weight, jammed and halted the whole run, frequently even passing under the bed.

Portable emery grinders were set up on the frames for cleaning the sheets, and the construction for the plate decks was changed. The installation of inter-roll slabs through the whole length of the run has been completed.



5) vertical stands ; 6) cold two-high stands; 7) universal four-high stands; 8) throttling installation; 13) disc shears; 14) drop-down axial shears; 15) sheet stacker; 16) continuous normalizing furnace;

The project also envisaged production of sheets with a rolled-edge. For this purpose a stand was set up with vertical rolls on a universal stand. However, despite a large number of experiments the rolled edge was not obtained, since compression in the vertical rolls gives rise to local thickening at the edges which flattens out again after passing through the horizontal rolls of the following stand, restoring the original shape.

One serious disadvantage was the small area of the sheet stacking bay and, most important of all, the poor maneuverability of the cranes in this bay. As a result, immediate arranging and handling of the finished product could not be achieved, thus holding back the productivity of the mill.

Individual shears, intended for shearing the sheets with deviations from the standard dimensions were badly located, at the opposite end from where the finished product arrived.

At the present time it is possible to utilize only one sheet stacker, and stacking of the finished product is tightly confined.

The project did not provide for mechanical take-off of samples for mechanical testing, which is very inconvenient. The question of stamping and marking the sheets was not considered before the end of the project stage. Up to the present time more than fifty workers have been employed on this section (not considering separating and stamping the samples). These operations are now mechanized and automatized. A rotating marking machine and automatic stampers have been installed at the mill.

The factory collective has developed a fairly stable roll profile for the finishing stand, making it possible to roll different profiles under conditions of extremely limited metal reserve. The mill now employs back-up rolls with concave profile and working rolls with alternating profile from cylindrical to concave with bending deflection of 0.3 mm.

All the supply lines of the mill have now been completely reconstructed (steam lines, air lines and lubrication system), having proved completely inadequate for winter operation as constructed according to the project. Considerable changes were introduced into the central, heavy and light oil lubrication units and also the hydraulic systems.

The fittings of the working stands were completely reconstructed including the driving beams and the hydraulic shock absorbers on the two-high stand, and the drives of the universal stand; the design of the bed rollers of the vertical stand was modified; the design of the bed rollers of the two-high stand was improved. The control system of the drive mechanisms was modified considerably; at the main control points, at the cutter disc control desks and several others. The construction of the stacking arrangements from the drop-down shears Nos. 2 and 3 was completely changed. Before reconstruction, the mill suffered considerable hold-ups since the scrap was removed manually. The design of the scrap-clearing machinery on the drop-down shears No. 1 was improved.

The factory collective, with the assistance of the laboratory workers, developed and put to work a new type of insulation (ceramic block lagging) for the feed pipes of the soaking pits.

In connection with a shortage in coke-oven gas the soaking pits had to be heated with oil according to the project. The oil consumption was 150 t per day on average. At the same time there was an excess of blast-furnace gas, which was frequently burnt at the blow-off pipes of the blast-furnaces. A system of combined oil and blast-furnace gas firing has been developed with success. This has reduced the oil consumption by 35-40%.

As a result of work carried out for improving the designs of the factory driving gear in the course of 7-8 months of operation the projected output was covered in a comparable grading and a start was made on fulfilling the plan. Since similar mills will be constructed at other factories in the U.S.S.R., planning organisations should study all the modifications carried out on the mill installed at the Voroshilov plant and take them into account in future projects.

## THERMAL INSULATION OF SKID PIPES IN HOLDING FURNACES.

V. V. Trofimov

(Engineer, Azovstal OTK Plant)

Modern holding furnaces fitted with skid tubes have definite advantages over furnaces in which the slabs and billets are moved along the hearth or the hearth beams.

Advantages of furnaces with skid tubes are low superheating, obviating the necessity for turning the billets; intense heating, reducing the time during which the billets are kept in the furnace; more uniform heating of the billets across the section; and saving in fuel.

Operating experience, however, shows that present furnace designs with skid tubes must be considerably improved in order to obviate the very large thermal losses.

The thermal losses can be reduced according to a proposal by the chief of the holding furnace section of the rail mill of our factory, F. V. Shilov, by fireclay blocks surrounding the surface of the skid tubes thus forming a thermal insulation. Circular skid tubes (with open surface) of 114 mm diameter and 13.5 mm wall thickness were replaced by tubes with a square cross section 142 x 142 mm and diameter of opening 65 mm. These tubes were very satisfactory. The square shape of the tubes provided a sufficiently reliable fixing for the insulating blocks.

The tubes were manufactured from manganese steel, with 0.30-0.35% C and 1.2-1.4% Mn content.

After assembling the blocks the upper section of the tubes was covered with a layer of fireclay paste. At the same time, the upper section of the transverse tubes, forming the supporting surface for the longitudinal tubes, was completely covered with fireclay. On the longitudinal tubes only the ribs at the top angle, serving as the skid surfaces for the billets as they are pushed forward, remained uncovered.

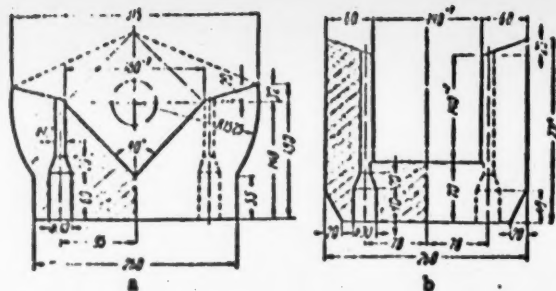
Use was made for the purpose of thermal insulation of fireclay blocks having the following chemical composition: 61.76%  $\text{SiO}_2$ ; 34.18%  $\text{Al}_2\text{O}_3$ ; 2.61%  $\text{Fe}_2\text{O}_3$ ; 0.73%  $\text{CaO}$ ; 0.4%  $\text{MgO}$ . The blocks (Fig. 1) are supported tightly one against the other along the entire length of the skid tubes and are clamped to the tubes by metal bars passing through holes in the blocks. The upper ends of the bars are bent outward and are shaped to the surface of the tubes.

The fireclay blocks have good mechanical characteristics: unit weight 1.88 g/cm<sup>3</sup>; after-shrinkage 0.21%; porosity 25.8%; and ultimate strength 238 kg/cm<sup>2</sup>. The fire resistance is 1670-1698°; temperature deformation, under a load of 2 kg/cm<sup>2</sup>, commences at 1370°.

Disadvantages of the thermal insulating blocks are the difficulty of fixing to the suspension bars and the necessity for additional insulation at the top (open) section of the tubes.

Despite this, the thermal insulation of the skid tubes provided positive results.

The holding furnaces at the "Azovstal" rail mill operate on a gas mixture (70% blast-furnace gas and 30% coke-oven gas). The mean calorific value of the mixture is approximately 2000 cal. The gas consumption was previously 30-31 million m<sup>3</sup> per month. Since introducing the thermal insulation on two furnaces (of the three operating) the gas consumption was reduced by 5.5 million m<sup>3</sup> per month.



**Fig. 1. Fireclay blocks for the thermal insulation of skid tubes by the suspension method:**  
a) for the longitudinal tube; b) for the transverse tube.

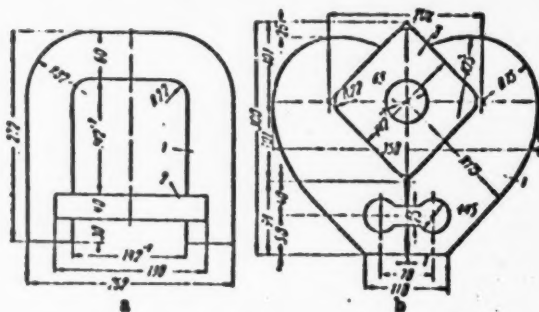
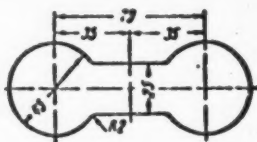


Fig. 2. Fireclay blocks for the thermal insulation of skid tubes by the suspension method;  
a) for the longitudinal tube; b) for the transverse tube;  
1) body of the block; 2) key; 3) skid tube.



**Fig. 3. Key for joining two halves of the insulating block for the longitudinal skid tubes.**

The reduction in the gas consumption represented a saving for the plant in July 1956 of 120,000 rubles and in August, when the thermal insulation was fitted to the second furnace, 193,000 rubles.

The time for heating billets from the cold bay was reduced by 20-25 minutes as compared with the previous program (depending on the profile and cross section of the billets). This is the more important since on these mills the cold bay holds 65-67% of all the rolled billets, and this reduction in the heating time provided a considerable increase in the productivity of the furnaces and rendered their operation with the mill smoother.

The quality of heating was also considerably improved. Before applying the thermal insulation the lower planes of the billets were not heated through so well as the upper. The temperature difference between the lower and the upper surfaces was 70-80°, giving rise to disuniform extension under compression in the mill and injuriously affecting the quality of the rolled sections.

This temperature drop was explained by the heat transfer from the lower plane of the billets by the uninsulated tubes. The difference in heating through was so considerable that it could not be obviated by maintaining the billets according to program in the soaking zone of the furnace. In individual cases the temperature of the lower plane of the billet taken out of the furnace was 100° lower than the top. On application of the thermal insulation, the temperature was equalized throughout the whole cross section of the billet. The bottom face was heated only 10-15° more than the bottom.

The uninsulated tubes did not usually last for more than 7-7.5 months, after which they had to be replaced by new tubes. The thermal insulation increases the life of the skid tubes by 2.5 years and more, thus considerably reducing the amount of difficult repair work.

Strict supervision of the heat technology is necessary. One serious infraction of the technology comprises melting of the slabs in the heating. In furnaces fitted with thermal insulation, melting is particularly intolerable, since the liquid slag falling on to the fireclay blocks disintegrates the surface and may result in untimely stoppage of the furnace for repair.

The plant collective is now working on modifying the shape of the fireclay blocks and substituting the metal suspension bars by interlocks made of fireclay.

In this way the blocks can be quickly and easily suspended from above. At the same time the surfaces of the skid tubes are completely covered by the body of the fireclay block (Fig. 2 and 3). As in the case of the first variant, the ribs of the top angles in the transverse tubes remain uncovered. Additional making-up with fireclay paste is obviated by this means. Preliminary tests of the blocks, as proposed, carried out on prepared models have indicated a high quality of assembly.

These methods may be varied for the purpose of further simplification in construction and greater reliability of fixing of the blocks. The principle of the proposed system of thermal insulation of skid tubes can be applied at other factories. With this system of thermal insulation, the quality of the heat is improved, the thermal losses are reduced, and the productivity and economy of operation of the holding furnaces are improved.

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## HOLLOWS ON WELDED TUBES

U. A. Mednikov

(Chelyabinsk Tube Mill Manager)

Hollows are the most frequently encountered defects on welded tubes. They occur both on the weld line and away from it.

Away from the joint the hollows may be formed through mechanical damage of the surface of the shaped piece in the first (shaping), second (welding) or subsequent reducing pass through pieces of slag or scale sticking to the rolls. In this case the hollows on the tubes have the appearance of spherical depressions or imprints (Fig. 1). If the surface damage on the piece has occurred through contact of the heated strip with projecting parts of the furnace, skid tubes or other details, the hollows have a drawn-out shape (Fig. 2). All the surface defects of the strip occurring in the course of manufacture naturally remain on the surface of the tubes. It is true that hollows occur much less frequently away from the joint than along the line of the joint.

If the edges of the strips do not coincide in the forming overlaps, twists, indentations and fissures occur on the welded tubes. These have the general name "joint hollows."

The occurrence of hollows along the joint depends not only on mechanical surface damage of the edges of the shaped piece and tube, but also on the welding temperature, spatial stressed state of the metal during shaping and rolling and the quality of the strip. Hollows along the strip joint (Fig. 3) may occur during passing through the rolls with adhering protuberances of slag or scale. In this case the edges of the strip, heated to a temperature near to the melting point, are readily deformed and do not coincide when forming the ultimate butt joint.

With an excessively high welding temperature the edges weld before the welding rod is applied. The fused droplets of slag and metal leave the surface of the strip, are driven off by the force of the compressed air jet onto the melted edges, turning them under (Fig. 4). If the temperature is too low, the edges are not sufficiently plastic and are deformed and turn inside the tube on making the butt joint under appreciable pressure.



Fig. 1. Spherical hollow away from the joint

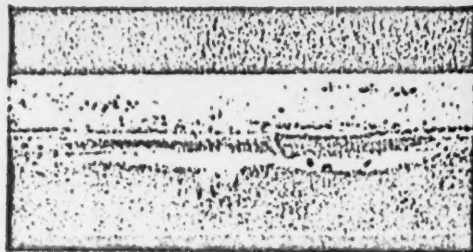


Fig. 2. Elongated hollow away from the joint

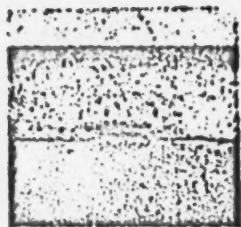


Fig. 3. Bending under the edges through scale on the rolls

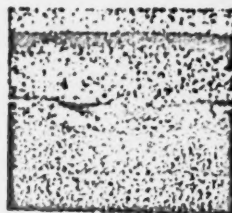


Fig. 4. Hollows through droplets of fused metal

In the case of disuniform heating of the two edges the colder edge will bend in to a greater extent than the hotter edge on forming the butt joint. Different heating of the edges frequently occurs as a result of disuniform blowing before the first pass.

One cause of hollows is also excess pressure in the welding gauge. The higher pressure is associated with excessive compression as a result of employing wider strips than provided for by the gauge, or incorrect drilling of the roll gauge.

In shaping tubes from wide strips (more than 150-160 mm) a combined nozzle is employed, consisting of a guide knife and nozzle tube. The guide nozzle is introduced between the edges of the sheet in front of the point of butting of the edges. Without this the edges of large strips overlap in shaping. However, with incorrect setting of this nozzle, fissures and hollows occur.

Hollows are also formed with displacement of the edges of the shaped strip in the vertical plane. This occurs particularly frequently on account of the presence of a gap (air space) in the shaft bearings, on which the shaping rolls are mounted. Hollows are frequently caused on tubes through poor-quality strip metal. The hollow locations are contaminated with nonmetallic oxides and sulfides and gas inclusions.

Hollows occur on welded tubes as a result of the particular conditions of shaping of the strip into the tube in the continuously rotating rolls. The occurrence of a large number of hollows in the shaping of the tubes is frequently associated with the presence of tensile stresses at the edges of the strips as a result of stratification of the metal.

Investigations carried out at the factory have shown that in order to avoid the formation of hollows it is necessary to apply careful supervision of the edges of the strip, discarding strips with edges displaying mechanical damage, obvious signs of stratification and other defects.

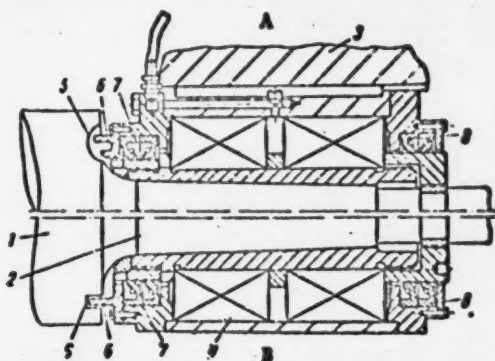
Damage to the edges of the strips must be avoided in the stage preceding their entry into the shaping gauges. Special attention should be paid to cleanliness and to the cooling of the first two shaping and welding gauges, and in the supervision of the operation of the blower equipment, together with the temperature regime.

## THE CHOCK PACKING FOR THE 300 ROLLING MILL

On the modern continuous rolling mills at the Makeyevka, Chelyabinsk and other plants the packing of the bottom chocks of the vertical rolls has a number of design shortcomings.

The rubber packing rings above and below the block are arranged in a similar manner (see diagram a). Hence, in delivering grease under pressure to the chocks at the bearings, the majority reaches the bottom packing. In the upper section of the roll through the semicircular groove with a large gap where the pressure flange is set up, water and scale enter, quickly wearing away the packing, and dropping through the chock onto the bearings, putting them out of action. The bearing is sometimes jammed on account of the scale, giving rise to breakage of the roll and of the bearing details.

Eight bearings were put out of action in a month on the B vertical stand roughing mill bay.



### Chock packing design:

- A) Before modifying; 8) After modifying;  
1) Roll; 2) Neck of the roll; 3) Chock; 4) Bearing;  
5) Semicircular groove; 6) Compression flange;  
7) Top rubber packing rings; 8) Bottom rubber packing rings; 9) Groove at right angles.

The design of the packing has now been modified somewhat (see diagram, b). The rubber packing rings have been set in a different way. A groove has been made in the rolls at right angles, and the ring compression flange fits tightly into the groove. As a result the grease does not pass downward but tends to go under the top packing (since these are set up in the path of the grease), fills the gaps in the groove with the flange and prevents water and scale entering the bearings.

With this packing arrangement the bearings have operated for four months without changing.

B. I. Lavrov (Chelyabinsk Metallurgical Section Mill Fitter Squad)

PLATE  
100

## PLATING AND GALVANIZING

### THE PREPARATION OF THE PROTECTIVE ADDITION FOR SHEET PICKLING

A. V. Kireeva, I. M. Kulikova, K. G. Plotnikova, and Candidate of  
Technical Sciences N. S. Smirnov

(Seversk Metallurgical Plant)

In order to obtain a clean sheet surface it is necessary to carefully prepare the protective addition before introducing it into the pickling bath.

In sheet mills the addition is usually prepared by the method of sulfuration, that is to say preliminary mixing with three times the quantity of concentrated sulfuric acid, so that the addition is more uniformly distributed in the pickling bath. Investigations have shown, however, that the coagulated ballast components of the addition, settling on the sheet surface, are one cause of bare spots in hot tinning. In order to remove the ballast components of the addition it was proposed that additional treatment should be applied to the addition in the form of settling and filtering after dilution. However, this did not completely obviate the precipitation of the ballast components of the addition on the sheet surface.

The black sheet addition—a product of oil distillation—consists of a number of organic substances. On treating the addition with sulfuric acid partial carbonization of the components occurs. The resinous ballast components are removed from the sulfurated component in the process of settlement and filtration and the fine carbonized particles pass through the top filter, contaminate the pickling bath and settling out on the sheet surface, form point contaminations. At a later stage (during tinning, galvanizing or other forms of treatment) they destroy the continuity of the coating.

In order to improve the sheet surface in pickling, an investigation was made of chlorination of the black sheet addition instead of sulfuration.

To one part of the black sheet addition three parts of hydrochloric acid were added (specific weight 1.17). After careful mixing, the temperature of the chlorinated addition was 40°, while with sulfuration the temperature was usually 125-150°.

Further treatment of the addition was carried out in the following way. After standing for a day, the scum formed on the surface of the chlorinated addition was removed. The addition was then diluted with a hydrochloric acid solution having a concentration of 150-160 g/l in the ratio 1:5. After careful mixing and standing for a day the scum again forming on the surface of the diluted addition was removed and the addition was filtered through a cloth filter.

In contrast to the very solid high-dispersion precipitate obtained on filtering the sulfurated addition, a flocculent readily filtering precipitate was formed on the filter.

The activity (protective quality) of the chlorinated and sulfurated additions was as follows, in %:

	Chlorinated addition	Sulfurated addition.
One hour after preparation	79.1	68.7
After dilution	81.5	85.9
After filtration	78.3	68.1

These figures show that the activity of the chlorinated addition is considerably higher than the sulfurated.

After pickling the sheet was annealed and sorted. The sorting indicated that the use of chlorinated addition had a very favorable effect on the surface quality of the pickled metal. Thus, the amount of sheet affected by black spots comprised 9.5%, while on metal pickled with sulfurated addition it amounted to 24.2%.

After complete sorting, the metal was subjected to polishing, repeat ("white") annealing and again sorted.

The data indicated that the quantity of sheet affected by contamination ("grease" and "black spots"), was much less when using chlorinated addition than when using sulfurated. Sheet suitable for tinning was pickled in a mixture of sulfuric and hydrochloric acid with an addition of sulfurated black metal addition and then tinned. Results of the grading of tinned sheet also indicated that much less contamination occurred on the sheet pickled with the aid of chlorinated addition than on metal pickled with sulfurated addition.

## NEW TECHNIQUES AND RATIONALIZATION

### For Discussion

#### MECHANIZATION OF THE LOOSENING-UP OF FROZEN ORE

V. I. Pereyaslav

(Section-Leader of the Department of Constructional Machinery TsNII, Mintransstroi).

and

U. A. Noskov

(Scientific Collaborator of the Railway Operations Department TsNII MPS).

Metallurgical plants in winter experience serious difficulties connected with discharging frozen ore, which is usually discharged manually from low wagons. Only on some large metallurgical plants is the ore discharged by means of wagon tipplers. The frozen ore in the low wagon must first be thawed or broken up.

In order to thaw the ore the wagon are run into a special heating shed. Heating the ore in the heating shed has a number of disadvantages, such as the very low efficiency of the heating shed as a heating device (11%), part of the rolling stock (particularly the braking system) perish, etc.

Breaking up the frozen ore by blasting is also not very effective in view of the limited extent of the work possible, damage to the rolling stock and high labor requirements for drilling the charges. Attempts to use vibrators for breaking up solidly frozen ore were also unsuccessful.

The All-Union Rail Transport Scientific Research Institute (TsNII MPS) and the All-Union Transport Construction Scientific Research Institute (TNIIIS) of the Mintransstroi have developed a mechanical method for breaking up frozen ore by means of vibration impact wedges (Fig. 1). The vibration impact wedge consists of a vibration hammer and sharpened steel rod, rigidly interconnected.

The vibration hammer providing vibration and periodic impact shocks to the rod consists of a bottom plate 1, a top plate 2 and suspension frame 3. The bottom plate of the vibration hammer takes the form of a steel construction with two brackets providing hinged connection of the suspension and four bolts for rigid connection with the rod wedge. There are also guide bolts 4 on the bottom plate for the springs 5 and two pairs of guide rollers 6. The hinges 7 connect the suspension with the brackets of the bottom plate.

The top plate of the vibration hammer takes the form of a shaped casting with two parallel horizontal cylinders onto which the stators of the three-phase synchronous electric motors are pressed. At the end of the rotor shaft of each of the electric motors, a pulley 8 is fixed with an eccentric load 9.

The dynamic action of the vibration hammer on the rod is produced by the fact that the rotors of the electric motors with eccentrically loaded pulleys rotating in different directions set up an alternating vertical centrifugal (agitating) force, causing oscillation of the top plate of the vibration hammer, which thus strikes on the bottom plate.

The rod is worked into the frozen ore and breaks off pieces from the main mass. The intensity of disintegration of the ore depends on the power of the electric motors of the vibration hammer, the weight of the impact mass, the centrifugal force, the total strength of the springs, and also the operating sequence of the vibration hammer. In addition, the dimensions and weight of the rod are important. In order to determine the influence of the shape and dimensions of the rod together with the sharpening angle, four types of equipment were tested (Fig. 2), the characteristics being set out in the table.

Tool (Fig. 2)	Point angle, degrees	Maximum cross section, cm <sup>2</sup>	Length mm	Total weight (with bearing plate), kg	Material
Rod diameter, mm:					
170 (a)	15	227	1100	140	St. 5
90 (b)	15	64	950	60	St. 3
H section wedge (c)	18	53	1000	65	St. 3
Three legged (d)	20	55	850	55	St. 3

The tests indicated that the most intense break-up of the frozen load is obtained with a rod of diameter 170 mm.

For the purpose of testing the efficacy of breaking up frozen ore and clay soil by means of the vibration impact wedge and also for clarifying the optimum working regime of the vibration hammer, the authors carried out a number of experiments. For driving the wedge into the frozen mass a type TsNIIS-7 vibration hammer was employed having two built-in electric motors type AB-52-4, each with a power of 7 kw. The main dimensions of the vibration hammer were 1150 x 1080 x 800 mm. The vibration hammer weighed 800 kg.

A special guide frame was employed for assuring a stable vertical position of the vibration wedge when driving.

By way of example, in carrying out one of the experiments an iron-ore sample 0.45 x 0.45 x 0.30 m with 0-120 mm fractions and moisture 9%, frozen at a temperature from -20 to -35° for twelve days was broken up with the vibration wedge in ten seconds. In the plane of the fracture, approximately twenty fracture locations of large pieces of ore were observed.

At the stock yard of one of the metallurgical plants, driving the wedge to a depth 650-750 mm into frozen chrome ore at a distance 400 mm from the edge of the dump slope produced fracture of pieces with dimensions on average 0.40 x 0.85 x 0.55 m, that is to say with a volume of 0.187 m<sup>3</sup>. The time taken for one driving was forty seconds, the specific energy consumption being 0.78 kwh/m<sup>3</sup> or 0.31 kwh/t.

It was established as a result of these tests that frozen dumps of ore, flux, and so on, could be effectively broken up by means of the vibration impact wedge.

For the practical application of the new method of breaking up frozen dumps, a special unit is necessary in the form of a portal, the inner contour of which is limited by the size of the rolling stock 2-B, and the outer profile by the proximity of structures along the run 2-C. The unit should travel along the loading-off bay on special rails set on the ends of the railway sleepers (Fig. 3). When idle the moving frame should be raised to the top position so that it will not interfere with the rolling-stock clearance.

The frozen load is broken up in the following way. The unit is brought up to the end low wagon and put into working position: the moving frame 2 is dropped so that the vibration impact wedges make contact with the surface of the frozen load. The protective vibration shields 8 are positioned so as to form continuation of the open hatches of the low waggon. The vibration wedge and vibration hammer drives are then switched in;

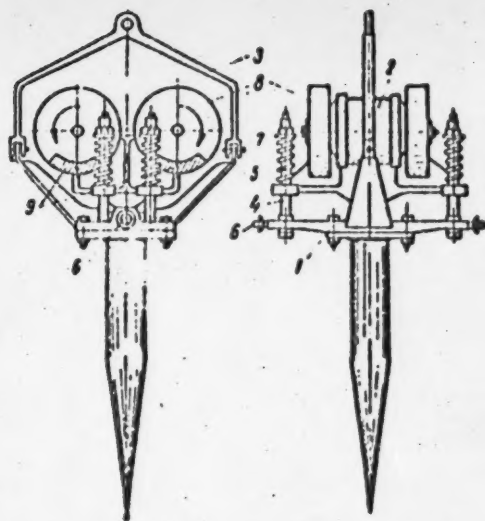


Fig. 1. Diagram of the vibration impact wedge.

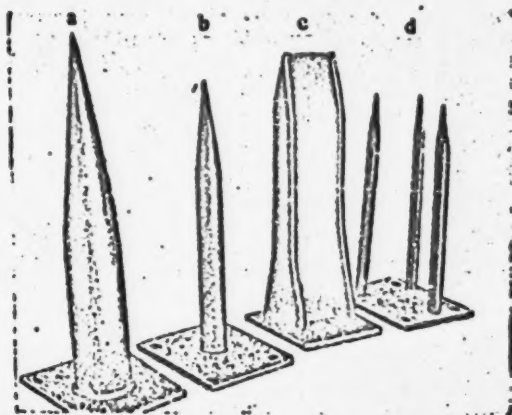


Fig. 2. Tools for breaking-up the frozen lead.

the wedges are driven into the mass of the frozen load. Subsequent double driving of the wedges, moving them along the wagon 200-250 mm breaks up part of the frozen load, which falls out through the hatch, forming an opening along the entire width of the open wagon. After this, the frozen load is broken up by consecutive fracturing of individual pieces from the main mass. Simultaneous driving into the frozen load of three wedges provides pieces which can fall through the hatch openings and slide down the slopes out of the range of the wagon.

On the basis of these experiments it can be assumed that for breaking up frozen ore in one low wagon thirty cycles of wedge driving will be necessary. This work will take, in all, thirty minutes approximately, including the time for moving the unit along the loading-off bay. A 50 kw supply point is necessary for operating the unit. A fitter and two assistants can service the unit.

Economic indices of the new method for breaking up frozen ore in off loading from the low wagon are difficult to determine accurately. Calculations carried out indicate, however, that the energy consumption for breaking up 1 t of frozen ore with the aid of the unit is only  $\frac{1}{34}$  th of that for heating 1 t of ore in the heating shed. For equal productivity of the heating shed and vibration impact unit the capital expenditure for setting

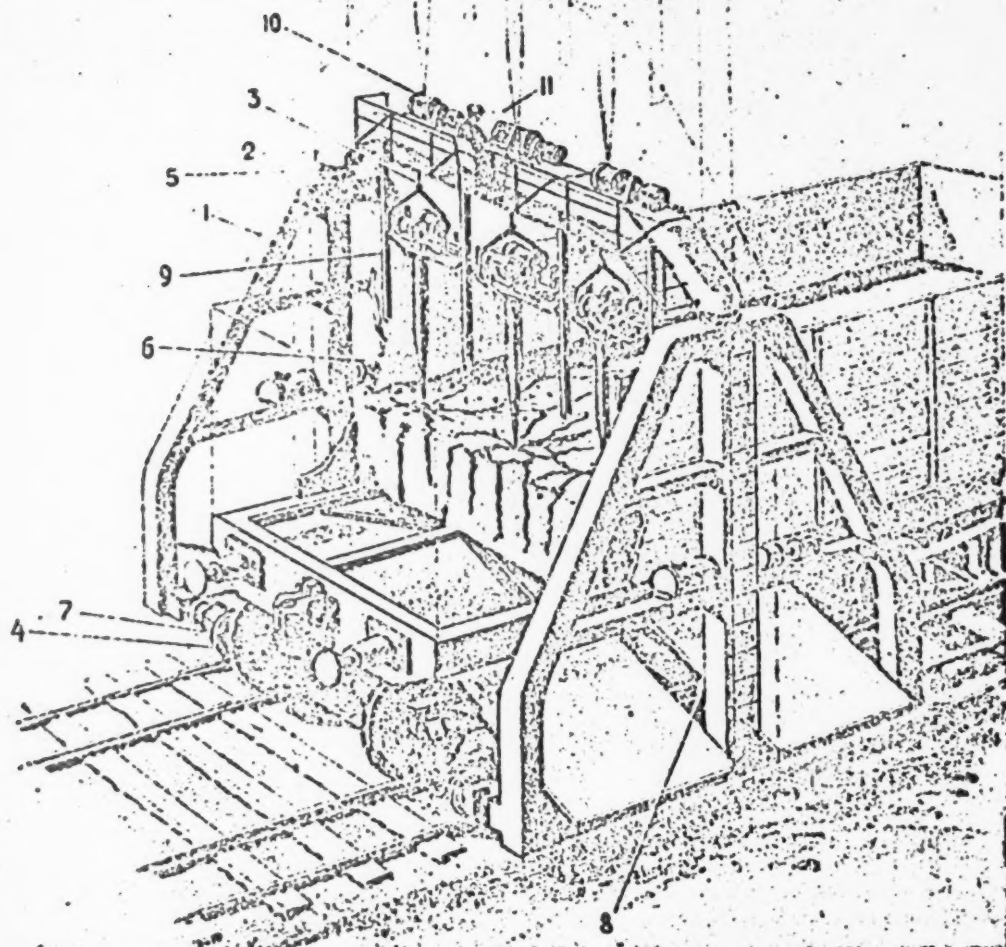


Fig. 3. Unit for breaking up frozen loads in low wagons.

1) Rigid frames; 2) connecting travelling frame; 3) vibration impact wedges; 4) wheels; 5) and 6) hydraulic cylinders with mechanism for vertical movement of the frame 2; 7) mechanism for driving the unit along the loading-off bay; 8) protective vibration shields, keeping the track clear of the discharged load; 9) vibration impact wedge guides; 10) vertical travel mechanism of the vibration impact wedges; 11) cabin with control desk.

up the units and the operating costs for maintaining them will be much lower than the equivalent expenditure on heating sheds.

The cost of manufacturing the first experimental model unit was in the order of 100,000 rubles. With mass production the cost of the unit would drop approximately to 50,000 rubles.

The use of vibration impact units would mechanize the work of breaking up frozen dumps in loading-off from the low wagons and reduce the idling periods of rolling stock on the metallurgical plant roadways.

BLANK  
3209

## CHARGING MACHINE FOR FEEDING MATERIALS ONTO THE HEARTH

F. F. Sviridenko and D. K. Kharlamov

("Azovstal" Plant)

In 1948 a charging machine was constructed at the "Azovstal" plant, operating on the principle of spraying the charge materials with compressed air.

Compressed air (5-6 atm) was delivered by means of a rubber hose to the air reservoir of the machine, made up from an oxygen bottle. A Laval nozzle was fitted at the head of the bottle, having an outlet diameter of 18 mm and tube diameter 100-120 mm, 3 m long, with an orifice to which the delivery funnel was attached. A bunker was set up above the funnel containing the charging material and fitted with a valve. In order to avoid blocking of the charging materials between the nozzle and the delivery funnel, injector tubes were attached along the sides of the bottle. The whole installation was mounted on a movable stand. Starting and stopping of the machine, together with regulation of the quantity of air delivered were carried out by means of a special valve.

On testing out the machine it was found that the charging material was not always carried over to the rear wall and was frequently blocked in the tube. This was partly explained by a drop in the compressed air pressure (below 4 atm) and the employment of dolomite of too coarse a grading.

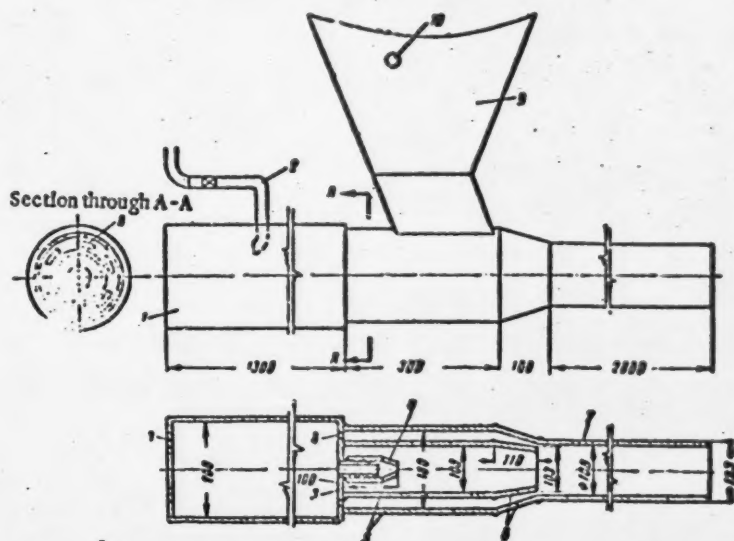


Fig. 1. Diagram of the pneumatic charging machine:

- 1) tube; 2) compressed air feed; 3) plugs; 4) Laval nozzle; 5) tube;
- 6) machined cones; 7) exhaust tube; 8) slots for air passage; 9) funnel;
- 10) trunnion.

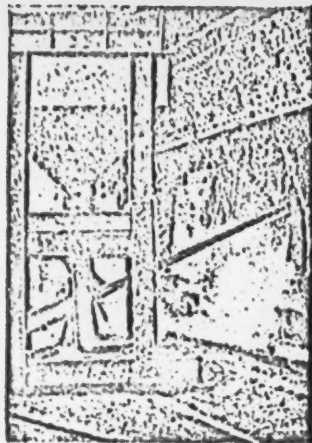


Fig. 2. General view of charging machine.

Repeat tests on the machine in 1955 showed that the machine in the proposed form, even with a higher pressure and using a standard grade dolomite, continued periodically to block up. The injector tubes did not overcome blocking of the material.

Improvements were introduced into the initial machine design, normally permitting utilization of the machine for mechanized charging of open-hearth furnaces and the delivery of filling materials onto the hearth during overhaul.

The pneumatic charging machine in the modified form is illustrated in Figs. 1 and 2. The machine is constructed from 200 mm diameter tube, to which compressed air is delivered by means of a rubber hose. Plugs are fitted in the ends of the tube, one which carries a Laval nozzle with an outlet diameter of 19 mm and two tubes with diameters 180 and 108 mm. Between these two small tubes the plug carries a number of slits for allowing the passage of air. Machined cones are fitted at the free end of the two tubes. The outer jacket finishes in an exhaust tube with inner diameter 108 mm.

A delivery funnel is fitted into the large tube, carrying trunnions by means of which the structure is fixed to the support.

In this form the machine is widely employed on the "Azovstal" open-hearth plant. Regulation of the machine is not difficult, the machine can be readily moved and provides a regular flow of charging materials. The machine is particularly suitable for overhauling the hearth. It provides a uniform thickness of charged layers and sets up a high density.

The average time taken for hearth overhauls in 1956 carried out with the aid of the compressed air charging machine was reduced by nearly five hours as compared with 1955.

## REDUCING THE CROP ENDS ON ROLLED SLABS

Candidate of Technical Sciences B. I. Panich

(Ukraine Institute of Metals).

In rolling killed steel slabs on the blooming-mill the fish-tail crop end comprises 2-4% of the length of the rolled slab. The quantity of fish-tail crop end is mainly determined by the depth of drawing on the bottom face of the roll, and the depth of drawing depends on the shape of the bottom section of the slab, i. e. on the configuration of the bottom of the moulds.

At present, in order to reduce the draft of killed steel slabs, the bottom of the moulds is given a spheroidal form. However, even this form of slab bottom-section (Fig. 1, a) does not reduce drawing to such an extent as to reduce the tail crop end to 1.0-1.5%.

A spherical form of mould bottom has been tested at the "Azovstal" plant for the casting of rail steel. Longitudinal sections of the fish-tail crop ends taken from slabs cast both in the usual way and into moulds with the modified bottom shape, indicated that with spherical bottom section shape of the slabs (Fig. 1, b) the drawing ("fish tail") is reduced on average to half. The fish tail was completely absent on some slabs. While in the transverse template cut at a distance of 1.5% of the length of the slab with usual bottom section shape of the slab, drawing traces were clearly visible (Fig. 2, a), in the transverse template cut at only 1.0% of the distance from the bottom face of the slab with spherical bottom shape, drawing traces were not observed (Fig. 2, b).

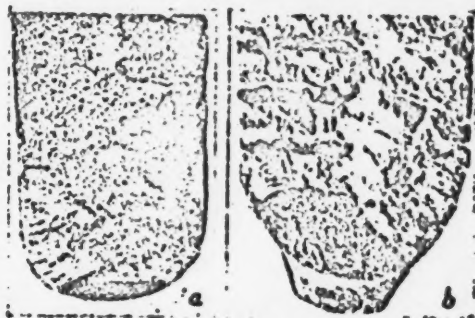
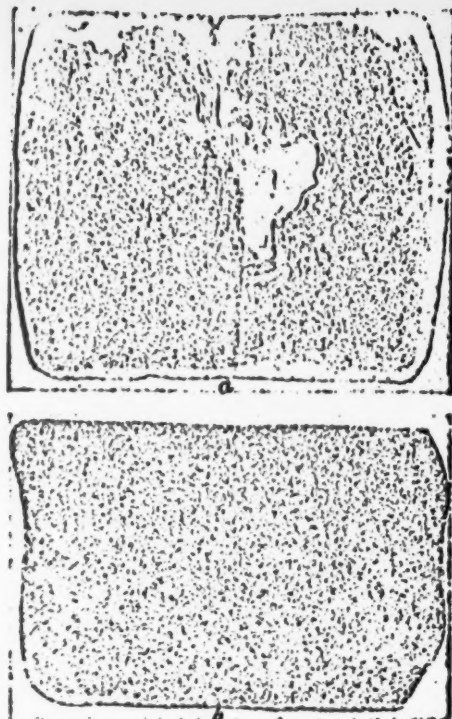


Fig. 1. Appearance of fractures of the bottom section of killed steel slabs:

a) usual spheroidal form; b) spherical form.

The introduction of moulds with spherical bottom shape at the "Azovstal" plant made it possible to reduce the fish-tail crop end to 1.0% of the length of the slab and to reduce the metal consumption coefficient in rolling rails.



**Fig. 2. Transverse templet cut from the lower face of rolled slabs:**

- a) ordinary slab (drawing traces clearly visible);**
- b) spherical form (drawing traces absent)**

Macroinvestigations on rails obtained from the bottom section of the slab showed a considerable reduction in the degree of damage to the surface of the test slabs from blisters beneath the crust. This is explained by the more rapid filling of the base of the moulds and the lesser possibility of welding of the metal spray with initial impact of the jet in the case of the moulds with spherical bottom shape (with top teeming).

## SCHOOLS OF ADVANCED EXPERIENCE

### STEEL TEEMING LADLES AND THEIR OPERATION

Engineers L. A. Volkov and N. V. Zaveryukha

(Magnitogorsk Integrated Metallurgical Plant)

In order to raise the productivity of existing open-hearth furnaces metallurgical plants are led to reconstruct the ladles for the purpose of increasing their capacity.

The weight of the metal construction of 200 t steel teeming ladles at present in operation is approximately 45 t and can be appreciably reduced, making individual units and details of the ladle simpler, more reliable and lighter, employing higher-grade metal and by changing over to welded construction. At the Zaporozhstal plant a 200 t ladle was designed the metal section of which was approximately 15 t lighter than the standard type ladle. The operation of this steel teeming ladle provides an increase in the steel production on existing open-hearth furnaces without re designing the pouring cranes.

The Inter-Works School for steel pouring, on visiting a number of metallurgical plants recommended the steel teeming ladle manufacturing plants to improve and lighten the metal section, to turn out complete ladles fitted with heat protecting shields for protecting the bottoms from excessive heating during steel pouring, with improved tilting fittings, stopper gear and pouring lips. The School recommended that other plants should take up the new solid stopper gear (Fig. 1) built by the Magnitogorsk metallurgical works, together with the new tilting gear cutting out participation of the workers in checking the ladle by means of the light hoist on the pouring crane.

The tilting mechanism operates on the following principle. The 45 t hoist on the pouring crane seizes the axle of the tilting gear. As the hook rises, the tilting gear automatically disengages from the hook attached to the ladle casing, and the ladle tilts. As the ladle drops from the inclined position to the vertical, the tilting gear automatically engages the hook.

In order to lighten the labor of the teemers and bricklayers and to improve removal of the slag from the ladles in pouring the smelt, the ladles at the Magnitogorsk metallurgical works are fitted with lengthened pouring lips as compared with the standard type, by which means it is possible to run off the slag directly to the slag bowls. This obviates the inconvenience attached to maintaining and overhauling of gutters and the bricklayer repairing the gutters had to carry out the work either at the lip of the ladle (standing under the gutter), or on a ladder set up against the gutter. The School recognized that this method of carrying-off the slag represents an improvement obviating the disadvantages of the old method still existing at a number of metallurgical plants. The extended lips are settled and dried on emergency stands under better and safer conditions.

The arrangement of the pouring lips differs at different plants. Some plants have ladles fitted with pouring lips both to the right and to the left. This allows the slag to be led off on pouring the smelt along a double trough on both sides into two slag bowls. At other plants, including the Magnitogorsk metallurgical works, the ladle pouring lips are arranged only on the right-hand side of the ladle. On pouring the melt on the double trough the slag is led off into the slag bowl through the pouring lip of the right-hand ladle, standing under the extended section of the branched trough. A minimum quantity of slag is allowed into the ladle standing at the center furnace.

In order to facilitate handling of the ladles, with pouring lips directed toward different sides, a number of plants have developed a ladle design with two pouring lips, on the right-hand and left-hand sides. In this case, however, it is necessary to extend the slag railway track into the bay.

Preparation of the ladle after pouring the metal is carried out by two different methods depending on the construction of the lining for the shell (lining brick or rammed lining). The treatment sequence of the ladles adopted at the Magnitogorsk works and adjudged by the Inter-Works School as the most efficacious has a number of advantages both with regard to the working conditions of the ladles and the teemers, and on the time expended for the preparation.

This is as follows. On completion of pouring from the ladle the liquid slag residues are poured off into the nearest slag bowl. The ladle is set up in a horizontal position on a special scaffolding, erected on the casting bay floor (Fig. 2). The ladle is cooled here for 2-3 hours by means of a portable fan (without the fan the ladle is cooled in 5-6 hours in the horizontal position and 6-8 hours in the vertical position on the stand).



Fig. 1. View of rigid stopper mechanism, set up on a ladle.

After cooling the lining, the edge, walls and pouring lip of the ladle are cleaned of encrusted slag. The slag cleaned off is collected in the ladle. The fixing nuts of the stopper mechanism are then unscrewed using a special key. In order to obviate untimely release of the stopper gear and endangering the ladle, special cables are fitted on the stopper gear before unscrewing the nuts. The washers are loosened with a crowbar from the supports on the forks of the stopper gears and the stoppers fall free onto the opposite part of the ladle wall. The stop-rods are now cleaned of the old stopper sleeves, withdrawn from the ladle by the 15 t hoist of the pouring crane and laid up on a special scaffolding.

Around the outside of the bottom a special table is set up from which the old shell is removed by means of a pneumatic ram and the lining brick is cleaned. The lining brick is well maintained and lasts for a long time. At the Magnitogorsk metallurgical works the life of the lining brick extends for more than five heats, and at the "Azovstal" plant eight heats, and so on.

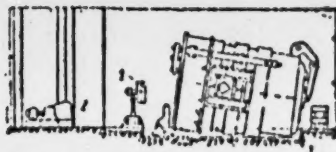


Fig. 2. Diagram of ladle position after pouring the melt:

- 1) portable fan; 2) electric winch;
- 3) special table.

Metallic scum weighing up to 2.5 t formed on the bottom of the ladle is removed by means of a special winch set up near the ladle. A 34 mm diameter steel cable is employed for hooking the scum. The winch is fitted with a 12 kw electric motor. The steel scum drawn out of the ladle is collected by means of the hauling crane onto a special platform. The use of a winch frees the pouring crane from the work connected with removing the scum from the ladle, the ladle treatment is considerably speeded up and the electricity consumption is reduced. Work on the ladle is rendered safer.

After finishing all the work of cleaning the ladle all the dirt is tipped out into a dirt box or slag bowl and the ladle is set up on the auxiliary stand. The bottom of the ladle is here repaired, the lining bricks and shell are set up, the lining is dried, the stoppers are fitted, the pouring lip is set right and the ladle is cleaned and blown with compressed air. Part of the work connected with workers being in the ladle is carried out with the aid of a special caisson chamber, set up in the ladle by means of the pouring crane.

The ladles set up on the auxiliary stands are adjacent to the working space of the furnace. The stands have ladders for mounting the pouring lip of the ladle from the furnace working area, fitted with upper and lower coke-oven gas burners. The upper burners are used for drying the ladle lining and the lower for drying the shell and brickwork. The brickwork is dried with a hood, so that the jet of the flame is directed at the setting location of the shell. The auxiliary stands are well lighted, particularly at night-time. Low voltage portable lamps are used for working inside the ladle.

At some plants the number of auxiliary stands fitted with burners is limited and the stands are specialized. On one stand the ladles are repaired, on another they are dried, and on a third the stoppers are fitted. With this type of work organization, the pouring cranes are occupied to a greater extent in transferring the ladle from one stand to another.

On comparing the expenditure of crane time in the treatment of one ladle, advantages can be seen in the method employed on the Magnitogorsk metallurgical works. At the NTMZ open-hearth plant, having five open-hearth furnaces, the pouring cranes were occupied on the preparation of one ladle for 75 minutes, while on the No. 1 Magnitogorsk metallurgical plant, having a miscellany of furnaces, the cranes were occupied in all for 35 minutes.

At those plants where the shell was set up in a rammed lining, another sequence of ladle preparation for the smelt is practiced. All the preparation work is carried out in the ladle standing in the vertical position. The ladle, freed from liquid slag residues, is set up by the crane on a shaft by means of which the old shells are removed, and the ladle is then set up on the reserve stand. The edge and walls of the ladle are here cleaned of adhering slag, the old stoppers are removed and the pouring lip of the ladle is cleaned. The dirt is tilted from the ladle and a caisson chamber is dropped into the ladle, by means of which (either manually or with a pneumatic tool) the lining is rammed and a new shell is set up. By this type of preparation of the steel teeming ladles the workers operate on the edge of the ladle in an unsatisfactory and unstable position at a high temperature.

At some plants, in particular at the KMK, the lining for the shell is rammed by means of a pneumatic tool, which provides better quality ramming. The hole in the rammed lining is cut out by means of a special knife. This lightens the work of the ladle workers. In contrast to other plants, at the KMK the mass for ramming the lining is prepared mechanically. The mass contains different grain sizes and the lining does not, therefore, show shrinkage in drying.

On setting up the shells in the rammed lining, the Podgorny device must be employed; on using lining brick it is sufficient to correctly set the lining brick itself and the shell itself is set up without the tool.

The School recommended all plants to work out charts of inspection and running repairs on ladles, to carry out a systematic technical documentation expressing the service of the ladles and, not less than three times during the year, to carefully examine the condition of the trunnions, bottoms and the side ring bands of all the ladles and to paint the outer surface with Kuzbas varnish instead of lime. Particular attention should be paid to greasing the trunnions and shafts of the tilting gear (or both hooks of the pouring cranes).

• NTMZ = Novo-Tashkent Metallurgical Plant-Publishers note.

•• KMK = Kuznetuk Metallurgical Combine-Publishers note.

The School considers it necessary to undertake standardization of the stopper gear arrangement on steel teeming ladles in such a way that the handle of the stopper mechanism comes off from the "cold" side of the teeming arrangement. In this case the teemer is near the empty mould, thus considerably facilitating his work during teeming. The stopper mechanism handle should be fitted to a screwed shaft (according to the Magnitogorsk metallurgical plant method) and made rotating, allowing the teemer to approach the ladle from any side. For fixing the handle it is better to employ slotted fingers and to fix them to the mechanism by means of chains; the locking nuts are fixed in the same way. This all facilitates the work of the teemers and expedites preparation for metal pouring.

## FIVE LOCOMOTIVES INSTEAD OF SIX

N. Podlesnova

(Transport Manager)

and

A. Brenner

(Work Organization Engineer, Novo-Tagil Metallurgical Plant)

The slag from the Novo-Tagil blast furnace is transported in three directions: to the grinding plant, to the cement factory and to the slag dump.

There is a double track between the points, with automatic interlocking; discharge of the slag to the users and to the dump is mechanized.

However, despite this, interruptions occurred on this stretch as a result of which the number of locomotives employed rose to six.

For the purpose of improving the work organization and establishing satisfactory operation of the section, a detailed study was made of all the work locations, and photographic timing was carried out on the operation of all the locomotives. The following shortcomings were detected as a result of the observations:

1. The system of payment of the locomotive crews was incorrectly drawn up, since the whole of the production of a shift was divided equally between the personnel of the six locomotives, and not according to the quantity of ladles carried by each locomotive.
2. The teams knew neither the standard time for delivering the slag, nor the time established for the run from the furnace to the point of slag delivery.
3. The absence of direct communication between the location of the cement factory station attendant and the cement factory resulted in discrepancies in the work of the attendant at the station, and the checker.
4. Poor lighting at night rendered the work of the team difficult.
5. The weight standard at the Shlakov-Vyazovka section was low.
6. As a result of the fact that the station attendants incorrectly distributed the slag, the ladles were frequently transported from place to place.

In the course of the examination, better levels of work were displayed by the checkers Zhigulev, Tsaregorodtsev, Ryzhkov and Mozhayev, engineer Grigoryev and assistant Kurochko.

The checkers applying advanced methods of work, taking over the shift, learn from the engineer about the state of coal and water on the locomotive, are informed by the station attendant as to the work in hand, receive concrete assignments for the immediate future and inform the whole team engineer, assistant and coupler of the plan of handling the work.

The checkers Tsaregorodtsev and Ryzhkov maintain close communication with the station attendant and blast-furnace worker, providing prompt information indicating to which furnace the ladles should be sent.

Ryzhkov at the cement factory post, immediately on receiving information checks the readiness of the section for delivering the slag to the user. While the slag is delivered he remains at all times near the locomotive and promptly resets the ladles. If there is a hold-up in delivering the ladles, Zhigulev and Ryzhkov immediately ascertain the cause of the hold-up and receive further information from the station attendant.

During loading of the slag, the free time is employed for stocking up the locomotive; the checkers assist the locomotive crew in this.

As a result of the examination carried out, measures could immediately be taken for obviating all the work irregularities exposed.

The system of payment was first re-examined. The payment for each locomotive-checker team was calculated according to the production coefficients as a function of the work involved in carrying out the operations.

This system immediately established a material interest. The teams started to work more productively. A hydrant was set up at the cement factory so that the locomotive could be topped up while delivering the slag. It was no longer necessary to spend time visiting the filling-up bay. Telephone communication was set up between the station attendant and the cement factory. All the crews were made familiar with the standard times for discharging the ladles and for the locomotive runs. A bogey chart was set up for delivering the slag to the destination. The railway track to the grinding plant was overhauled; on some sections the necessary number of brake chocks were installed. The weight standard for slag delivery to be dumped was increased from four to five ladles.

For the purpose of disseminating the experience of the best checkers and engineers, schools of advanced work methods were organized.

As a result of introducing the improved methods of work, five locomotives instead of six now serve the slag side of the furnace continuously and strictly to program. The annual saving through introducing this measure amounts to 320-360 thousand rubles.

## REFRACTORY BRICK-CUTTING MACHINE

In repairing metallurgical furnaces a large amount of dressed brick is necessary. Until 1955 the brick was dressed manually at the Dzerzhinsky plant.

In 1955 the maintenance fitters P. T. Orzul and F. A. Svistun put forward a proposal for reconstructing the machine erected earlier for cutting refractory brick. They replaced the flat belt drive with a V belt, for the cutting tool employed an electric carborundum wheel on a bakelite disc type LT-3, K-16, STB, grade 50 (outer diameter 400 mm, thickness 4 mm) and changed the design of the work-bench. After these modifications the machine was put to work and is operating reliably. The machine is extremely simple in construction (Fig. 1 and 2) and can be set up on any plant or construction site.

The frame of the machine is welded from angle-iron, to which the lever mechanism and tilting cable are affixed. The electric motor (type AO 41-4, 1.7 kva., 1420 rpm) and the electric carborundum wheel are mounted on the tilting table.

A 190 mm diameter pulley is mounted on the motor shaft; a 165 mm diameter pulley is mounted on the electric-carborundum stone shaft. The electric carborundum wheel drops down when the foot presses the working pedal, while the work-table moves forward with the brick fastened to it. The brick is dressed to the necessary size.

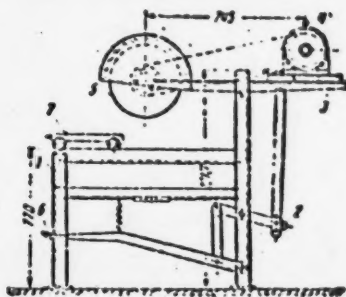


Fig. 1. Diagram of refractory brick dressing machine:  
1) welded frame of the machine;  
2) lever mechanism; 3) tilting table;  
4) electric motor; 5) carborundum wheel; 6) operating pedal 7) work-bench.

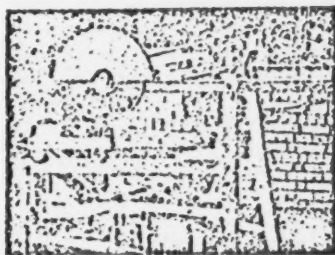


Fig. 2. General view of the refractory brick-cutting machine.

The machine has made possible the utilization of cut-off sections of refractory in the overhaul. The productivity of the machine amounts to 300-350 pieces of standard magnesite brick, or 100-150 pieces of side brick. One wheel on average is expended for every 150 dressed bricks. The cost of a wheel is 4 rubles 50 kopeks.

The use of the machine for dressing brick has provided a saving on the plant of 50,000 rubles per year including 20,131 rubles saved by using 60% of all the off-cuts in the structure.

N. P. Borody, Engineering Workshop Rate Setter,  
Dzerzhinsky TRMP Works.



